Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine.

Vol. IX | New York, December, 1911. | No. 12

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

J. W. Richards, Ph.D., President.
James H. McGraw, Vice-President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
239 West 39th Street, New York.
Telephone Calls, 4700 Bryant.

Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

J. M. Muir, Manager.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs.)

Copyright, 1911, by the Electrochemical Publishing Company.

Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879.

New York, December, 1911.

CONTENTS.

Editorial:
Chlorination versus Cyanidation ... 619
Spelter and Natural Gas ... 620
The Electric Furnace and the Scrap-Metal Business ... 620
American Institute of Chemical Engineers ... 622
Surface Combustion ... 622
The Western Metallurgical Field ... 622
Non-Ferrous Metal Market ... 623
The Iron and Steel Market ... 623

Correspondence:
Manufacture of Pig Iron in the Electric Furnace. By John J. Smith ... 624
Abandoning the Terms Carbonic Oxide and Acid. By Carl Hering ... 625
Induction Furnace Design. By Albert Hiorth and F. A. J. FitzGerald ... 625
An Instructive Experiment on the Flow of Heat. By B. S. Lacy and Carl Hering ... 625
Presidency of the American Electrochemical Society ... 626

Sizing, Diametric and Volumetric With and Without Screens. By B. F. Rice ... 627
Flameless or Surface Combustion. By John B. C. Kershaw ... 628
Direct Smelting of Argentiferous Copper Ore at Rio Blanco, Chile. By I. D. Ossa ... 630
The Electric Reduction of Iron Ores. By Otto Frick ... 631
Remelting of Ferromanganese in the Electric Furnace and the Use of Molten Ferromanganese for Dioxidation. By F. Schroeder ... 640
The Production of Oxygen in the Workshop. By Aug. S. Neumark ... 641
The Application of Electricity in the Metallurgical Industry in Italy. By R. Cantani ... 642
Apparatus for Making H_2S. By J. R. Huber ... 649
Platinum Laboratory Utensils. Committee Report of Am. Ch. Soc'y. ... 649
Flow of Heat Through Bodies. By Carl Hering ... 652
Iron and Steel Institute Meeting. (With Abstracts of Papers) ... 654
Continuous Carbon Dioxide and Temperature Recorder and Its Application to Combustion Efficiency. By Ed. A. Uehling ... 656
Notes on Chemistry and Metallurgy in Great Britain ... 658
Recent Chemical and Metallurgical Patents ... 659
Synopsis of Metallurgical and Chemical Literature ... 671
Induced Drafts for Sulphuric Acid Chambers ... 674
Centrifugals for Laboratory Works ... 675
Repair of a Large Rock-Crusher Shaft ... 676
Frictionless Ink-Type Recording Instrument ... 677
Electro-Bessemer Furnace ... 678
Automatic Controller for Motor-Driven Machinery ... 679
Census Report on Sulphuric, Nitric and Mixed Acids ... 680
Continuous Carbon Dioxide and Temperature Recorder ... 681
Welding. (Meeting Am. Soc. Mech. Eng.) ... 682
New Detinning Works ... 682
Fixation of Atmospheric Nitrogen in Norway ... 682
Notes ... 682
Personal ... 683
Digest of Electrochemical and U. S. Patents ... 684
Book Reviews ... 684

Chlorination versus Cyanidation.

Less than a decade ago the respective merits of the chlorination and cyanide processes of treating gold-bearing ore formed the basis of many a heated controversy which aligned leading metallurgists in two opposing factions. Each process had its champions, but neither held undisputed sway in practice or received the unqualified approval of a majority of the profession. To-day cyanidation appears to have the field to itself, only one prominent chlorination plant remaining in this country, viz., that of the United States Reduction and Refining Company in Colorado.

With apparently such a sweeping victory for cyanidation it might be surmised that the disparity between the two processes is greater than it really is. An excellent opportunity has been afforded in Colorado for a close comparison of the two processes as applied to Cripple Creek ores. This comparison is the more complete because both chlorine made by electrolysis and chlorine generated from bleaching powder and sulphuric acid have been used for years in two competing custom plants. It is a significant fact that the one using electrolytic chlorine continues to treat ore by chlorination, while the last to use bleaching powder and acid has abandoned the method in favor of cyanidation.

In the matter of initial cost of installation of plant the chlorination process is more expensive than cyanidation, due largely to the lead-lined solution tanks, etc., required in the former process. If electrolytic chlorine is used, another item also is added to plant cost. Again, the chlorination process is restricted in its application to non-argentiferous gold ores and to those containing little or no lime. In the latter respect chlorination has been losing ground in Colorado, as lime has been an increasing constituent of Cripple Creek ores, requiring the addition of as much as 40 lb. of sulphuric acid per ton of ore treated.

The cost for chemicals in the two processes will be about the same per ton of ore treated, provided sufficiently cheap electrolytic chlorine is available for chlorination. If chlorine can be made at a cost of four cents per pound for salt and power, the latter costing about one cent per kw-hour, it is likely that chlorination can compete with cyanidation in cost of treating a favorable ore. Under the conditions given the cost of chlorine would range from 18 to 30 cents per ton on sulpho-telluride ores such as are now being mined in the Cripple Creek district.

In refining costs chlorination has an advantage over cyanidation, as it also has in the possibility of making monthly cleanups, thereby avoiding the necessity of carrying a large amount of gold in solution to be estimated in the monthly inventory. This is considered quite an important point in favor of chlorination for a custom plant.

Obviously the chlorination process has lost ground for good economical reasons. But it is a fact that in none of the chlorination plants of the past or present the latest advances in electrolytic chlorine processes (which have been significant in more than one way) have been employed, simply because they were not yet available when those plants were built. With electrolytic chlorine now constantly looking for new outlets, it is not impossible that the pendulum may swing back and chlorination may get another and better chance in the metallurgy of certain particularly suitable ores.

Spelter and Natural Gas

One of the most striking examples of the attractiveness of cheap fuel in metallurgical enterprises, is to be found in the Kansas-Oklahoma natural gas fields and the large zinc smelters located there. Statistics on the Western production of spelter show the transference of the zinc smelting industry from the vicinity of coal fields in Illinois and Missouri to the newer and more attractive gas fields of Kansas and Oklahoma. But no doubt with the gradual exhaustion of natural gas we shall witness a migration of the smelters back to Illinois.

In the first eight years of the decade from 1901 to 1910 the production of spelter in Illinois remained almost stationary at from 45,000 to 50,000 tons per annum. This increased slightly in 1909 to about 67,000 tons, and again in 1910 to 73,000 tons. The Missouri production during this decade held a fairly steady average at about 11,500 tons for the first eight years, but fell to 7000 tons in 1909 and 5500 tons in 1910. During the same period, however, the spelter output in Kansas increased from about 75,000 tons in 1901 to 105,000 tons in 1910. And beginning with 1907, which was the first year in which Oklahoma was credited with spelter production, that State has increased in output from 5000 tons to 35,000 tons in 1910.

This change in the reduction center of the zinc industry has been due to the cheap natural gas available in Kansas and Oklahoma, and any future change to Illinois will be due to cheap coal, for the spelter producing States are but minor producers of zinc ore, while the largest ore producing State is a diminishing producer of spelter. When the production of spelter in 1910 was apportioned to States according to source of ore, Kansas was found to have produced but 4.05 per cent of the spelter made in the United States; Illinois, 0.61 per cent; Oklahoma, 0.91 per cent, and Missouri, 55.71 per cent. But when apportioned to locality in which the ore was smelted, regardless of its source, Kansas led in spelter production among the Western States, followed by Illinois, Oklahoma, and Missouri in the order named.

The natural gas in Oklahoma and Kansas is nearly pure methane, containing approximately 90 per cent CH_4, from 1 to 2 per cent CO, and the balance oxygen, nitrogen, and carbon dioxide. The heat value of methane is 966 b.t.u. per cu. ft., and of carbon monoxide, 344. Based on the analysis of the natural gas it has a heating value of about 875 b.t.u. This is a high value compared with that of producer gas, which averages 150 to 160 b.t.u. per cu. ft. Natural gas is now bought for smelting in Kansas and Oklahoma for three cents per thousand cubic feet, which is equivalent to producer gas at 0.6 cent when fuel values are considered. Prices for other industrial uses of natural gas in this region are higher than for smelting, the enormous volumes used in the latter work giving the advantage of a lower price. For firing boilers in the Joplin field, to which district a great deal of gas is piped from Oklahoma, gas costs 12.5 cents per thousand cubic feet, while for gas engines the cost is 25 cents. Electric power generated from gas is sold at Joplin at a cost of one cent per hp-hour, and can be generated by individual large plants at a cost of six to seven mills per hp-hour.

In view of the favorable natural resources of the Kansas-Oklahoma-Illinois district it is not surprising that ore and concentrates from Montana, Utah, Colorado and other Western States, as well as from Mexico, are shipped there on favorable freight rates for the long haul. But the prospect of exhaustion of the gas is always in sight, and even now the tendency is to raise the price. Kansas has declined as a producer, but Oklahoma is rapidly coming to the front and promises large volumes of gas. Seven cents per thousand cubic feet is now being asked for some industrial uses, and even higher prices will prevail where the gas has to be piped long distances.

The Electric Furnace and the Scrap-Metal Business

We have pointed out on several occasions in these columns that the electric smelting and refining furnace has several peculiar fields of its own, due to the differential advantages which its own nature gives it. Chief among these fields is the electric refining of a semi-refined steel producing a high-grade tool steel, which has had a considerable degree of commercial success in the past few years. Then the production of the difficultly reducible ferroalloys, such as ferrosilicon, ferrotitanium and ferrochrome, has been an assured commercial success both here and abroad for some time past. At present we see the electric brass furnace and the electric smelting furnace for zinc-lead ores both measurably close to the commercial stage. In all these three are certain innate points of superiority, or, to be concise, certain points of differential advantage in the application of electric heat to the work to be done.

These same advantages repeat themselves in the case of the melting down, reducing and refining of metal refuse or waste products from cities and manufacturing plants. The first point of superiority that electrothermic treatment possesses over the older forms of treatment is high degree of control over temperature. This is apparent even to the layman, and it is both true and humorous that many engineers are more or less laymen when it comes to electrometallurgy. Granted once the proper reciprocal relation between the generation of heat and the absorption of heat we have as a logical conclusion perfect control over the temperature of the reacting zone and over the operations therein. As a matter of practical fact, in any successful electric furnace we have an approximation to this ideal condition.

Next in apparent importance is the control over the chemical properties of the fluxes and slags and the reducing properties of the atmosphere. This is particularly true of the electric zinc furnace. Other advantages are absence of dust and heat, the low labor charge and the freedom from exacting care by the furnace men. It is really surprising to see an electric furnace in operation with pointers on the ammeter and voltmeter staying at their positions "as if they were glued there." Such is the great regularity of electric smelting when it is properly

understood. As the thermal efficiency of the electric furnace is always high, provided it is over 150 kw capacity, all these factors combine together to make a low working charge with a high degree of metallurgical economy.

The greatest advantage to our way of thinking is a commercial one—the low capital cost provided power is bought from the outside and only the transformers and switchboards are owned. As the energy transferred per unit of working volume is large—3 kw to 10 kw per cubic foot—the furnace is not expensive.

So we see that every item of reasoning points directly to a wide potential field for the electric furnace in the smelting and refining of scrap metal and waste products that our foundries and manufacturing centers produce. The commercial reasons are both simple and obvious. In any large city many tons of such stuff as brass-founders skimming, skeleton brass, steel borings, etc., are produced each year. In any large city there is always a surplus of electrical power that can be sold reasonably on the 24-hour basis for nine months and on the 20-hour plan during the months of December, January and February. Join these two industrial things by a successful electric-furnace process and we have the raw material which is produced locally turned into the finished product locally. Four handlings and two railroad hauls are avoided. Besides there is created a market for some of the surplus power of the central station. With such a powerful metallurgical instrument at hand, the losses of metallic values would be necessarily low if this same agent were intelligently applied. In general there seems to be no room for doubt that the electric furnace will find a permanent useful place as a means for working up waste material into available material. In this century elimination of waste equals efficiency.

Of the various specific instances that seem apparent, the reduction of skimmings from brass or bronze foundries strikes us as the most promising. These analyze from 40 per cent to 70 per cent zinc plus tin plus copper. Consider a "government or navy" bronze of the composition copper 88 per cent, tin 10 per cent, zinc 2 per cent, which sells in the ingot form for 15 cents or 16 cents. The dross from this will contain a greater proportion of the more easily oxidized tin and zinc. It is sold for its copper contents alone at from 3 cents to 4 cents a pound. Experiments of which we have definite knowledge show that it is not a difficult task to reduce this in an electric furnace and with the mixture of copper to produce an ingot substantially the same as the original alloy. How much better it would be to have this done than to have the skimmings bought by a copper smelter where the copper alone is recovered and that, too, in the debased form of copper-matte. To repeat, the place to do the converting is at the point of production. Naturally there are obvious difficulties. Chief of these is the chance that there would be contamination of skimmings by iron or other foreign substances. It may also be argued that there will be a displacement of the crucible by the electric alloy-furnace which makes no dross and that hence the supply of skimming will rapidly diminish. But practically this should not carry too much weight because human inertia will see to it that the crucible furnace will be only gradually displaced by the electric alloy-furnace. The electric lamp has not supplanted the gas lamp, nor has the latter kerosene lighting. Indeed some wiseacres say that the consumption of candles per capita is as large as ever.

The next apparent fissure in the existing commercial structure into which the electric furnace can insert its wedge is the melting down of steel borings, chips and turnings. The same remark can be made of similar materials of cast iron. Their bulky nature makes them expensive to transport and smelt. The electric furnace which has no blast to oxidize the great surface exposed or to blow them out of the stack has a marked natural advantage over the open-hearth furnace in the case of steel borings, while it has a slight though marked advantage over the cupola furnace in the case of cast-iron borings. In each of these instances the proper and logical procedure is to take the borings, chips and turnings which are commercially unavailable, practically speaking, and to turn them electrically into steel castings or iron castings which are commercially much desired and of great worth. A fact that is apparent and of importance is that it is simple to attain in the electric furnace a casting of good physical properties, *i.e.*, freedom from slag, blow-holes and dissolved oxides. Interdependent with this is the fact that good physical properties conduce far more to a sound casting than do good chemical properties. Naturally, there are many essential details of engineering and business that must be learned and mastered in order to make this a success. But the manufacture of the finest and most difficult steel castings or iron castings out of the cheapest kind of scrap is not only theoretically possible but actually has been done. How attractive it is commercially can be seen by the fact that the raw materials sell at present for from 25 cents to 40 cents per hundred pounds and the finished material sells for as high as 6 cents a pound or over.

Other specific possible examples can be adduced, such as the smelting of tin dross, which indeed is being accomplished commercially, or of Babbitt metal dross. The melting of aluminium dross is possible, but here it is not possible to reduce the oxide, but only to save the entrained prills of metal. The reduction and distilling of galvanizer's skimmings and the galvanizer's "dross" is also open to the electric furnace. Unfortunately, the problem here is complicated by the presence of sal ammoniac in case of the skimmings and the necessity for a practical condenser in both cases. The refining of plumbiferous refuse products is also feasible. But here the older types of furnaces are securely intrenched. But the distinguishing characteristic in many of the above instances is that no great inventive ability and no great business enterprise is needed. A combination of average metallurgical experience and of average business knowledge and leavened by a grain of originality is enough.

A certain amount of commercial inertia, of course, acts like a brake on this and on all material changes in our industrial structure. Indeed it is very proper that it should be so, even though the adventurous minded think it unnecessary. But we are firmly of the opinion that in every manufacturing center good opportunities exist for the "little fellow" in electrothermal treatment of waste metal material, for the capital cost of a five-ton electric smelting and refining plant should not exceed $20,000. Broadly speaking, both specific and general observation indicate that the "trusts" leave plenty of just such opportunities for the "able little fellow."

American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers will be held in Washington, D. C., from December 20 to 22. The headquarters will be at the New Willard Hotel, where all the professional sessions will be held.

The first session will be held on the morning of December 20 (Wednesday), when the following papers will be read and discussed:

Clarence Hall, "Advances in Testing Explosives."

Richard X. Meade, "Distribution of Power in Portland Cement Manufacture."

J. T. Baker, "Manufacture of C. P. Acids."

L. S. Hughes, "Combustion of Pulverized Coal."

Ludwig Thiele, "Manufacture of Gelatin."

S. F. Peckham, "The Natural Bituminous Rocks of the United States."

In the evening the retiring president, Dr. F. W. Frerichs, will present his address on some problems in chemical engineering practice (manufacture of chloroform from alcohol; construction of laboratory apparatus; manufacture and testing of shipping cylinders for liquid ammonia gas).

This will be followed by a paper by John C. Minor, Jr., on the manufacture and testing of carbonic acid cylinders.

On the morning of December 22 (Friday), there will be a symposium on the U. S. patents, papers to be presented by Messrs. E. B. Moore, Edw. T. Fenwick, Walter D. Edmonds, and R. N. Kenyon.

In an evening session Mr. F. G. Wheeler will present an illustrated lecture on the adaptation of the centrifugal pump to chemical service, and Dr. J. C. Olsen will read a paper on a course in chemical engineering education.

The social functions include a subscription dinner at the New Willard on Thursday evening and a reception at the White House on Friday afternoon.

The program of visits and excursions includes the Bureau of Standards on Wednesday afternoon, the U. S. Proving Grounds at Indian Head, Md., on Thursday, the Patent Office on Friday afternoon, and the steel plant at Sparrows Point, Baltimore, or the cement plant of the Tidewater Portland Cement Company on Saturday.

Surface Combustion.

A most interesting lecture was presented on October 27, by Prof. William A. Bone, of the University of Leeds, Eng., at the Chemists' Club in New York, before the members of the different chemical societies. Professor Bone gave a review of the results of the work which he has done on this subject for a long series of years, and his discourse, as well as the very successful experiments by which the lecture was illustrated was greatly enjoyed.

The subject is fully covered in a special article by Mr. Kershaw on page 628 in this issue.

The Western Metallurgical Field.

Cyaniding Concentrates.

A fine example of careful experimental work, followed by successful application of the results obtained, is to be found in the cyaniding of concentrate at the Treadwell mines, Alaska. At the time the experimental work was undertaken the concentrate was being shipped to the smelter at Tacoma, and the cost of treatment of 3-oz. gold concentrate was $11.95 per ton. From the experimental work it was estimated that the concentrate could be treated at home by cyaniding at a cost of $3.25 per ton. The details of experimental work and the present operation of a 100-ton plant were presented by Mr. W. P. Lass, superintendent of the plant, at the recent San Francisco meeting of the American Institute of Mining Engineers. We note here the most interesting features of the experimental work, which will show the development of the process now used.

The concentrate from the various mills amounts to about 1.8 per cent of the original ore, and contains: Fe, 40 per cent; S, 1 per cent; SiO_2, 11 per cent; Au, 2.5 to 4 oz. gold, and 0.75 oz. silver per ton. Preliminary tests were made in glass jars, after which 50-lb. composite samples were ground to 200 mesh in a clean-up barrel with iron balls, the pulp passed over amalgamated plates and then agitated in small Pachuca tanks 4 ft. high and 14 in. in diameter. The results showed that 75 per cent of the gold could be recovered by fine grinding and amalgamation, and 96 per cent by amalgamation followed by cyaniding. The next step was the construction of an addition to one of the mills, and the erection of a 4-ft. x 12-ft. Abbé tube mill. Various forms of classifier were tried, the Dorr proving most satisfactory. Callow cones were used for dewatering, and the agitation was carried on in a Pachuca tank 10 ft. in diameter and 22 ft. high.

The results further showed that by grinding in a 1.5-lb. cyanide solution an extraction could be obtained by cyaniding alone equal to that obtained by cyaniding and amalgamation, or 96 per cent. Thus a satisfactory extraction was obtained either by amalgamating and cyaniding or by cyaniding direct. A preliminary agitation with alkali solution was found to shorten the time of cyanide treatment and save 25 per cent in cyanide consumption. Passing the air used for agitation through a receiver filled with caustic soda or milk of lime also decreased the cyanide consumption, presumably by the removal of oil and CO_2 from the air. When grinding in solution of over 1 lb. per ton, followed by amalgamation, it was difficult to keep the plates bright, due to a dull white surface deposit which, if allowed to remain, turned to a dull gray. A muntz-metal plate was substituted for a copper plate, but as all the plates were coated, no variation in the result was noted, and there did not appear to be any essential advantage in its use.

The results obtained from a period of two years' investigation justified the building of a 100-ton plant. The plans provided for the use of amalgamation if desired, and ultimately nearly 5000 tons were treated by each method before finally deciding to abandon the time-honored amalgamated plate.

The following table shows the work of one of the 5-ft. x 22-ft. tube mills when treating an original feed of 88.75 tons per day.

	+ 100 mesh %	+ 200 mesh %	— 200 mesh %
Feed	48.7	41.5	9.8
Discharge	10.1	26.4	63.5

This product is classified in Dorr classifiers and only the overflow passes to the cyanide tanks. This material screens more than 98% through 200 mesh.

An interesting device is used in the Pachuca tanks to bring settled concentrate into circulation. After the first treatment the pulp is allowed to settle, and it forms a 30-ton packed mass in the bottom of the Pachuca. This is brought into agitation in 15 minutes by means of a device known locally as a spider, which is an adjustable hollow annular casting with radiating fingers, the whole encircling the central agitation column. When the charge is to be put in agitation the spider is lowered by a small hand windlass until it rests on the settled charge. Air is then turned through the fingers, and at the same time the solution for the next treatment is run into the tank. The device rapidly bores its way to the bottom of the Pachuca, clearing a path to the bottom of the central agitation column. As soon as this is opened and air has been admitted to the central pipe the spider is raised from the tank and full agitation of the charge proceeds.

Kelly presses are used for the final filtration of the pulp, and Merrill zinc dust precipitation presses are used for the recovery of the gold precipitate. In connection with zinc dust precipitation considerable annoyance has been occasioned by the clogging of the cloths in the gold presses. Filter cloths of several kinds have been tried, and now the lightest and cheapest muslin is used with results no worse than with the more expensive

grades. Another trouble was experienced due to the accumulation of precipitated gold in the pipes carrying the gold solution and zinc emulsion to the presses. The interior of the pipes gradually becomes smaller in area; in one instance a 6 in. pipe having an area of 28 sq. in. was reduced by incrustation of gold slime to less than 4 sq. in. After three months' use 80 ft. of the pipe was cleaned with a recovery of $25,898.

A change has been made in the Merrill precipitating presses, so that the effluent solution passes through open pipes into a launder on top of the presses instead of through cocks as usual. This was to permit of intermittent operation of the presses without draining the solution each time the press was cut out, and thereby allowing zinc to oxidize and the mass to settle in the frame so that the full efficiency of the zinc could be obtained. This change has resulted in raising the gold value of the precipitate from $15 to $25 per pound.

The cyanide plant has now been in operation for a year, and during the month ended August 15, 2010 tons of concentrate were treated at a cost of $2.8764 per ton and an estimated extraction of 97.025%, as compared with the experimental estimates of $3.25 per ton and 96% extraction.

New Smelter for United Verde.

The United Verde Copper Mining Company, Jerome, Arizona, is to build a new and modern copper smelting works. The United Verde has long been a prominent copper producer, largely owned and controlled by W. A. Clark. Owing to the policy pursued by the owners, little has been known about the operations of the company, either financially or technically, and this lack of knowledge has caused many rumors to spread regarding the company. Metallurgically speaking, the United Verde was for many years in the fortunate position of being able to run and make money without due regard to that careful technical direction which is considered essential by our present large producers. Many interesting stories are told of the perplexities of former managers and superintendents, many of whom were better fitted for many other forms of work than smelting; and whether true or not, they illustrate well the ludicrous situations in which a non-technical man might find himself in conducting the smelting business.

A number of misfortunes have befallen the United Verde, such as mine fires and caving ground, the latter having been specially disastrous because the smelting plant had been built on the mine property and the settling of the ground damaged the buildings and necessitated much reconstruction. It has been rumored repeatedly that on account of this condition the company would erect a new plant, and undoubtedly the present decision has been influenced by the misfortunes of the past. It will be remembered that the United Verde expected to make a trial of the Knudsen converting furnace, as reported in these columns some months ago, but this was rendered impossible by reason of a serious cave which ruined the construction work before the process could be tried out.

The new smelter will be situated about six miles from Jerome, and at a lower elevation than the present plant. The men in charge of construction are C. H. Repath and J. B. McIntosh, who were prominently identified with the designing and construction of the Tooele plant of the International Smelting & Refining Co., in Utah.

Tooele Lead Plant.

Construction work at the lead plant of the International Company has been progressing, and all of the excavation and foundation work is reported to be completed. Some of the minor buildings such as the bag house, extension of power house, flues, etc., have been completed, and a large amount of steel is on hand for the construction of the bins. Erection of the furnaces is to proceed at once, and it is hoped that two of the three furnaces will be ready for operations early in 1912. New ore contracts are being made with Idaho and Utah producers, and apparently there will be a large quantity of ore on hand and contracted for when operations begin.

Nevada Hills Mill.

In October the new mill of the Nevada Hills Company, situated at Fairview, Nevada, commenced operations, and is now running at practically full capacity. The plant should be one of the best in Nevada for its construction and design were in charge of engineers who had been connected with other successful plants. According to preliminary reports the mill has shipped its first consignment of gold and is making a recovery of between 92 and 93%.

Company Reports.

Utah Copper Company report for the third quarter of 1911 shows that 1,273,373 tons of ore were treated at both plants, the average grade of ore being 1.4829% copper. The gross production for the quarter was 25,851,456 lb. copper, an average monthly production of 8,617,152 lb. The average cost of copper per pound, after making allowances for smelter deductions, and without crediting miscellaneous income, was 7.56 cents, the lowest in the history of the company. The entire Magna plant was in operation through the whole quarter, but only five remodeled sections of the Arthur plant were in commission. The Magna plant treated about 12,000 tons per day and the Arthur plant 3000 tons. This indicates a final capacity for the Arthur plant of 8000 tons per day when the thirteen sections are completely remodeled.

The Ray Consolidated Copper Company has issued a report showing operations since the interim report of April 1, 1911. Four sections of the concentrator are in operation, running at a capacity of about 800 tons per day each. The sections were designed to handle 1000 tons each daily, and have demonstrated their ability to do so. The gross production is about 2,500,000 lb. per month from ore averaging 1.85% copper. The mill extraction is averaging 69%.

Non-Ferrous Metal Market.

There has been little or no change in the market for non-ferrous metals since our last report. The demand for all continues good and the prices generally are firm, reflecting the active condition of foreign markets.

Copper.—Domestic consumption has not been more than normal, but the demand for export has continued to be heavy. As a result there was a large business at advancing quotations. At our last report Lake copper is quoted at 12⅜ to 12⅝ cents, and electrolytic at 12.30 to 12.35 cents.

Lead.—The market is unchanged at 4.10 to 4.12½, St. Louis, and 4.20 to 4.25, New York.

Tin.—A large business has been done in this metal, mostly for future delivery. Stocks are so low that a premium on spot tin may come to pass. November tin was quoted at 42¼ cents per pound.

Spelter.—The scarcity of November spelter was more pronounced than in previous months, but there has been no change in the prices. Business has not been more than normal for immediate needs. St. Louis spelter was quoted at 6 to 6.10 cents and New York at 6.15 to 6.25 cents.

The Iron and Steel Market.

November in the iron and steel market opened with gloom universal and closed with sentiment uniformly optimistic, buying heavier and prices showing a tendency toward firmness.

Sentiment had been uniformly pessimistic for three months when on Oct. 26 the United States Government brought suit against the United States Steel Corporation for its complete dissolution, not simply a dissolution of the Steel Corporation as a holding company, but a dissolution also of the component elements, the consolidations of 1898, 1899 and 1900, which in 1901 were brought under the ownership of the corporation. The immediate effect of the bringing of this suit was a fresh accretion of pessimism, but this made no material

impress upon the market, which was depending upon actual conditions and not upon sentiment for its strength. So thoroughly had the previous pessimism liquidated the situation that the market rather tended to improve.

Suddenly the railroads became rather free buyers. Several large purchases of rails for 1912 delivery were made, and although rails were the only commodity which had suffered no decline, the $28 price established early in 1901 having been maintained since, no question seems to have been raised, and the orders booked, aggregating 200,000 tons or more, appear to have been placed at this price, although perhaps the sales were made with a guarantee against decline. It has long been known that the railroads are much more interested in raising the standard of quality than in depressing the price.

In the closing days of October roads comprising the New York Central system began placing orders for cars, and other car orders soon appeared. The total bookings for November cannot be stated at this time, and the movement is not yet over. In brief, it may be said that in the first half of this year car orders amounted to about 50,000 and in the next four months to about 20,000, while November and December are likely to see a total of between 50,000 and 100,000. When early orders in this movement amounted to 20,000 the large plate mills estimated that the business would be sufficient to give them as heavy operations through December as they had had in September and October; later orders guarantee them heavier operations than formerly.

An incentive for the purchase of cars at this time is their cheapness, prices being lower than at any time since the steel car came into vogue. The railroads, however, are believed actually to need the cars. They have never been known to make large purchases in cars or any other line merely because prices were low, even though it has frequently been urged that in certain cases they would eventually save money by doing so.

Unfilled orders of the United States Steel Corporation increased during June, July and August, decreased 84,668 tons in September and increased 83,011 tons in October. Estimates are that in November a considerably larger increase occurred.

Production of pig iron in November was at a slightly lower rate than in October, when the rate was approximately 25,000,000 tons a year, but there had been a continued increase since July.

Operations in the steel industry as a whole continued at about 75 per cent of full capacity. It should be observed, however, that at the present price level no more than 90 per cent could be expected to be operative, so that the addition of business equal to 15 per cent of capacity would in all probability effect some slight advance in prices.

The general level of finished steel prices, averaging the various commodities approximately according to their tonnage importance, declined about $1 per ton during November, as was the case in both October and September, while the months of July and August together witnessed a decline of about $1 a ton.

Pig iron production in 1911 may be forecasted at about 23,700,000 tons, to fulfill which prediction would require about 370,000 tons more to be made in the second half than the 11,665,706 tons officially reported for the first half.

Pig Iron.

The pig iron market has shown remarkable quietness. Almost as a unit buyers have refrained from making any purchases for the new year and are endeavoring to clean up their yards completely by the end of this year. Purchases have been very light. On the whole, the statistical position of the furnaces is good, as in the main they are reducing stocks. The present situation is fraught with some danger, in that buyers may wait too long and then be forced to come into the market in a body, with the result of prices advancing and an unwarranted increase in production occurring. The most important transaction in November was the purchase of 25,000 tons of Bessemer iron by the Republic Iron & Steel Company from the Ohio Iron & Steel Company, operating a merchant stack at Lowellville, near the company's Bessemer steel plant at Youngstown, Ohio. The price paid was $14.25 and delivery was to be completed in a few weeks, the iron being shipped from stock. Brokers had sold a few odd lots of Bessemer at $14, a cut price, and the Republic purchase served to stiffen the market. The market stands quotable as follows: Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2X, $14.50 for basic, $20 for standard low phosphorus; f. o. b. Chicago district furnaces, $14 for No. 2; $14 f. o. b. valley furnaces (90 cents higher delivered Pittsburgh); Bessemer, $14.25; basic, $12.50; No. 2 foundry, $13.25; forge, $12.75; malleable, $12.50 to $12.75.

Steel.

The market in billets and sheet bars has been a decidedly open one, with special prices made on the few transactions which have occurred. The general level may be quoted at $20, Pittsburgh, for Bessemer and open-hearth billets and sheet bars. Rods are quotable at $26, Pittsburgh.

Finished Material.

Open quotations on bars and plates declined $1 a ton during the month, and on wire products and structural shapes $2 a ton. At the present level the market is firmer all along the line than it has been at any time for months. There may be some further declines, but in general the present level is likely to hold, and some slight advances may occur before the end of the year. This is the first time since April that an advance in any steel commodity is a distinct possibility.

We quote regular open market prices as follows, these prices being occasionally shaded:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.10 to 1.15 cents, Pittsburgh.

Shapes, 1.10 to 1.15 cents, Pittsburgh, for beams and channels, 3 to 15-in., inclusive, zees and angles 3 x 3-in. and larger.

Steel bars, base, 1.05 to 1.10 cents, base, Pittsburgh, for prompt delivery, 1.10 to 1.15 cents on contracts for first quarter; 1.20 to 1.23 cents, Chicago, 1.10 to 1.15 cents, Buffalo.

Iron bars, 1.25 cents, Pittsburgh; 1.20 cents, Philadelphia; 1.15 to 1.20 cents, Chicago.

Wire nails, $1.55, base; plain wire, 1.35 cents, base.

Black sheets, 28 gage, 1.85 cents; galvanized, 2.85 cents; blue annealed, 1.40 cents; painted corrugated roofing, $1.35 per square; galvanized, $2.50, f. o. b. Mahoning valley mills.

Tin plates, $3.40, Pittsburgh, for 100-lb. cokes.

CORRESPONDENCE.

Manufacture of Pig Iron in the Electric Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the paper of Mr. E. Odelberg in your October issue on electric pig iron and the manufacture of steel therefrom much stress is laid on the fact that when a low-silicon, low-carbon iron is produced which is filled with blow holes the inner surfaces of these blow holes are white, showing entire freedom from oxidation (page 509).

It is pointed out that an entirely different condition prevails as to the surfaces of blow holes in similar iron made in the blast furnace, this showing all the colors of the rainbow; the oxide colors familiar in hardening and tempering steel. This difference is held to prove the entire absence of an oxidizing condition and of oxides in the hearth of the electric furnace and their presence in that of the blast furnace.

This would be interesting and important if it were true, but as a matter of fact, the writer has made many tons of white iron full of blow holes (which is known under the trade name of "Spongy Six") in a charcoal blast furnace, and of the many thousand blow-hole surfaces examined, none ever showed an

color except silvery white just as described by Mr. Odelberg.

The real truth of this matter is that these blow holes are produced by the action of imperfectly reduced ore in the bath reacting on the carbon (presumably that dissolved in the iron) and forming thereby carbon monoxide. This, of course, is unable to exert any oxidizing influence under the conditions prevailing in the bath, and the interior of the blow holes which it forms are necessarily free from oxide and perfectly white in consequence. The carbon monoxide does not escape from the iron because the temperature prevailing in the hearth under such conditions is extremely low and the iron is too pasty to permit the bubbles of gas to rise through it.

The electric furnace will come into its own in time, probably in Sweden and on our Pacific Coast sooner than elsewhere on account of the presence in those regions of iron ore and water power in abundance, and the scarcity of mineral fuel. But it is as well to let its growth be a natural one, unhindered by extravagant claims which tend to make the claimants, and sometimes the process, ridiculous.

As the ores of iron are all oxides and gradually reduced as they descend through the furnace, it is always possible for cold or irregular operation to introduce into the hearth imperfectly reduced material containing oxygen, quite irrespective of whether the source of heat for the reduction process be the combustion of carbon or an electric arc. This is so rudimentary that it should not require statement, but apparently it does.

JOHN J. SMITH.

Abandoning the Terms Carbonic Oxide and Acid.

To the Editor of METALLURGICAL AND CHEMICAL ENGINEERING:

Sir: Some time in the middle of the last century chemical nomenclature was reformed by progressive chemists, so as to make terms correspond more nearly with the facts; the formula for water, for instance, was thus changed from HO to H_2O.

It was, I believe, at that time that the consistent applications of the terminations -ous and -ic were introduced for the lower and higher oxides respectively, in order that the names should correspond more nearly with the compounds. According to this nomenclature the lower or monoxide of carbon, CO, should for consistency sake be called carbonous oxide, and the higher or dioxide, CO_2, be called carbonic oxide, even though there may still be some uncertain theories about the valency of carbon.

Notwithstanding these very desirable reforms, and the length of time since they were introduced (more than a generation ago), one still finds the term carbonic oxide used by some for the *lower* oxide CO, the higher one being called by them "carbonic acid," although it is in no sense an acid, any more than the oxide of sulphur, SO_3, is sulphuric acid. These were the older terms used before the reform of the progressive chemists. A prominent metallurgist even went so far as to say that he had never heard or seen the term carbonic acid used for anything but CO. I believe it has been the custom of careful writers for many years to abandon the older nomenclature.

At present the state of affairs is that when one hears or reads the term carbonic oxide one is at a loss to know what the author meant, unless the context determines it. In view of the fact that in metallurgical reactions it generally makes a very great difference whether CO or CO_2 is meant, this confusion is most unfortunate, and as it seems to take more than a generation to bring about a complete reform in terms, and that it therefore seems hopeless to do so at the present time, it would, in the opinion of the writer, be better to abandon the term carbonic oxide entirely (and of course also the positively incorrect term carbonic acid for CO_2) and use only the terms carbon monoxide and carbon dioxide, about which there can be no doubt; this I believe is what is taught in modern up-to-date colleges. It would be better still to use the symbols CO and CO_2.

There is no objection to using the term carbonous oxide for CO, as there should be no confusion. After several generations have passed by the term carbonic oxide might perhaps again be introduced so that the nomenclature concerning carbon will be consistent with that used for other elements, like cuprous and cupric, ferrous and ferric, stannous and stannic, etc.

Philadelphia, Pa. CARL HERING.

Induction Furnace Design.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the report of the Toronto meeting of the American Electrochemical Society, in your October issue, page 536, Mr. FitzGerald is reported to have said, in reply to an inquiry of Dr. Waterhouse, that the Roechling-Rodenhauser furnace for such sizes would be found greatly superior to other forms of induction furnaces.

As I and certainly many other readers of your journal would be greatly interested to know *in which respects* a Roechling-Rodenhauser furnace of this size would be greatly superior to mine and other induction furnaces, I ask you to put this inquiry to Mr. FitzGerald into your columns.

Kristiania, Norway. ALBERT HIORTH.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: Unfortunately I have not got a copy of the stenographer's report of the discussion following Mr. Hiorth's paper at the Toronto meeting of the A. E. S. My remembrance is that Dr. Waterhouse asked for someone's opinion, specifically mentioning my name. I think my reply was substantially as reported, but it must, of course, be treated as an opinion based on my own experience, *so far as it goes.* This is a different thing than an opinion based on an investigation made for the purpose of comparing all types of induction furnaces. I think it is only fair to Mr. Hiorth and his furnace to explain this.

My reply to Dr. Waterhouse was based on the experience I have had in the construction and running of various induction furnaces, on what I have seen of the running of an 8-ton Roechling-Rodenhauser furnace and on what I have read of other induction furnaces, such as that of Mr. Hiorth. There can be no doubt at all as to the superiority of the R-R furnace over various induction furnaces I have had personal experience of. I have seen the R-R furnace running in a most satisfactory way on cold scrap and hot metal from Bessemer converter and turning out high-quality steel.

As to the Hiorth furnace—so far as I know the largest one that is regularly working has a capacity of 5 tons, and according to Richards' report last year[1] was melting high-grade materials used in the manufacture of metal of crucible-steel quality. According to the same authority[2]: "At Mr. Hiorth's plant the induction furnace is used for *melting down high-quality metal* and making a high-quality steel, *but he also uses an electrode furnace for working up scrap.*"

The italics are mine. So far as my information goes the R-R furnace seems up to the present to be the best tool, working by induction, for the manufacture of steel.

FRANCIS A. J. FITZGERALD.

FitzGerald and Bennie Laboratories, Niagara Falls, New York.

An Instructive Experiment on the Flow of Heat.

To the Editor of Metallurgical & Chemical Engineering:

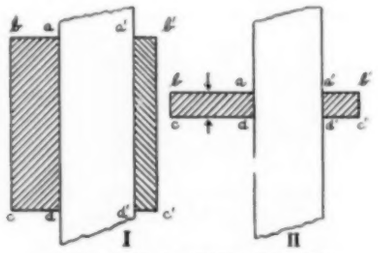
SIR.—In the communication from Mr. Carl Hering in the November issue, entitled "An Instructive Experiment with the Flow of Heat," some conclusions are drawn which rest on a misconception of the conditions involved.

Without going into the mathematics of the conduction and

[1] Transactions of A. E. S., vol. 18, p. 200.
[2] *Metallurgical and Chemical Engineering*, Oct., 1911, p. 545.

radiation of heat, it should be pointed out that Mr. Hering is not warranted in arguing from the fact that more heat is lost through the long than through the short end of a long metallic *rod* held in a flame, that, therefore, more heat will be lost through a thick wall than through a thin one.

Now the cooling in the former case takes place not merely at the ends of the rod, but also—and here to a far greater extent—at the cylindrical surface; while in the case of a furnace or a "tank containing a red hot liquid," the *wall* may be considered as an *extremely short* thick rod of which the cooling surface consists to all intents and purposes of the end only, the remaining surface being comparatively small. In other words, the radiation conditions in the wall are exactly the reverse of those in the rod.



CONDITIONS OF WALLS AND ROD.

In Fig. I, which represents the wall conditions, the total radiation surface is practically the same with the thick wall as with the thin one, being composed chiefly of the end surfaces of which *b c* and *b' c'* are the diameters. Consequently the amount of heat lost will depend on the rapidity of the *conduction* in the walls, which will evidently be lower for the thick one.

In Fig. II, which represents the rod, on the other hand, the cylindrical surfaces of rotation about *a b* and *a' b'* make up practically all of the total—and in this case it depends on the relative magnitudes of the radiation and of the conduction coefficients of the material whether more heat will be lost through the long or the short rod. Thus, with a good conductor like copper, Mr. Hering finds the long rod loses more heat; with a poorly conducting substance, like pipe clay, the reverse seems to be true.

The important practical conclusion, however, which needs to be emphasized here, is that noted above, namely, with any material, a thick wall will insulate better than a thin one. Thus, Mr. Hering's experiment does not contradict furnace practice.

B. S. Lacy.

Perth Amboy, N. J.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: Mr. Lacy would do less injustice to others if he would read over more carefully what he criticises with such rather strong terms as "misconception," "not warranted," etc., before publishing such criticisms.

If he will read over my communication again, he will find that the chief point in it (which he seems to have missed) was to call attention to the surprisingly high virtual thermal resistance which exists at a red-hot surface exposed to the air. In this there are no "misconceptions of the conditions involved"; at least he has not been able to point out any.

In his second paragraph he misrepresents me if he is trying to make it appear in the last clause as though I had concluded that the loss through thick furnace walls in general was greater than through thin ones. The term copper is not mentioned by him. What I did say applied only to copper walls, and I distinctly stated that with clay, and therefore probably with other refractories also, the results would very likely be reversed.

I did and still do maintain, and Mr. Lacy will probably not deny it, that with such a monstrosity as a furnace with relatively very thick walls of copper, the leakage per sq. ft. would be greater through the walls than through the top surface of the charge, if liquid. This would certainly be true when the cross sections and exposed surfaces had the same relations as in the experiment, and this is all that I claimed. Such a monstrosity was used by me only as an illustration in an argument, not as a practical case. When the walls are relatively thin, I should want to try an actual experiment before drawing conclusions. The mathematical relations are complicated and the final result depends on several relations combined (see this journal April, 1911, p. 189). Until Mr. Lacy can show proof to the contrary, I therefore deny that I am "not warranted" in drawing the conclusion which I did.

Mr. Lacy's diagrams are not very clear, but as I understand them he has omitted the flare of the walls, which, of course, is one of the chief features implied by the term relatively thick walls. A furnace with relatively thick copper walls is, of course, a practical absurdity, and was used by me merely as an extreme case, in order to bring out the point more clearly.

In his last paragraph I differ with him when he says "any" material, as with relatively thick walls of copper his conclusions are not correct. In this paragraph he again misrepresents me, if he intends to imply by it that I had claimed that my experiment contradicted furnace practice. My original article clearly states that with clay the results would probably have been the opposite to those with copper, and therefore would confirm furnace practice. In fact, it was to try to satisfy myself that thick walls of insulating material were justified, that I made the experiment.

I desire to add here, however, that I believe that increasing the thickness of walls may easily be overdone, and my experiment tends to strengthen this view. Our data of the necessary physical constants is too meager to determine this mathematically. A furnace which I recently designed remained positively cold on the outside, and it is therefore a question whether the extra thickness was warranted. I am inclined to believe that the thermal resistance of fire-brick work is higher than we now suppose, probably owing to the joints, particularly when the bricks are set parallel to the walls so as to make as many joints as possible, as was the case in this furnace.

Philadelphia, Pa. Carl Hering.

Presidency of the American Electrochemical Society.

To the Editor of Metallurgical and Chemical Engineering:

Sir: As the Eighth International Congress of Applied Chemistry will be held in this country in the year 1912, on which occasion many foreigners are expected to visit this country, and as electrochemistry and metallurgy are represented as sections of this congress, the American Electrochemical Society should, in our opinion, take this into consideration in the selection of its president for the coming year, as he will be expected to represent this society at the official and social functions.

Dr. Charles A. Doremus is in our opinion especially well qualified for the presidency for that year, and would do credit to the society. We hereby recommend to the members of this society that they nominate him in the ballots which they will soon receive. He has an international reputation as a chemist, is a graduate of Heidelberg University, under Bunsen, is versed in the duties of international functions, is acquainted with many of the foreign visitors, having been Official Delegate from the State Department to the Second International Congress, and made chairman of the American Committee of Organization by Moissan, for the Fourth Congress, is conversant with the three principal languages, is able to devote the necessary time to the duties which would be imposed upon him, has been a Director and Vice-President of this society, a member since its organization, was one of the eleven original organizers taking a very active part in starting the society, and in the work of the New York Section, of which he has been Chairman.

Charles S. Bradley, Henry S. Carhart, Carl Hering, P. Heroult, Alois von Isakovics, W. R. Whitney.

Sizing, Diametric and Volumetric, with and without Screens.

BY B. F. RICE.

In a large number of the manufacturing industries and in the treatment of ores it very frequently becomes necessary to grade dry and granular material according to size or bulk of particle. This constitutes one of the most serious problems with which engineers and mill men have had to deal.

Up to the present time machines working on two separate and distinct principles have been used for this work. I refer, first, to the different forms of classifiers which classify the material according to its weight, putting a large particle of low specific gravity with a smaller particle of high specific gravity. This is accomplished by the introduction of an hydraulic or pneumatic floating medium. A grading of this kind can be made also on dry material without the use of a floating medium. A machine for work of this character was developed by us, but was discarded for want of a promising field.

Second, different forms of screens, such as the impact, shaking, belt, inclined, trommel and gyratory screens, which have met with only a limited degree of success. This is probably due to a number of different causes, but more particularly to the high cost of maintenance, together with a rather low efficiency.

It has been our aim in the development of the McKesson screenless sizer[1] to provide a cheaper and more efficient method of sizing dry granular material than is afforded by any screening device. This was no small undertaking, and as we were the originators of the idea, as is clearly shown by the United States patent records, we have not only developed a new machine but an entirely new principle, which appears to the uninitiated to be contradictory to the known laws of gravitation. But like a great many other radical departures from old established laws, it is very simple when once understood.

Our policy has been to bring this machine to such a high state of development that its commercial success would be assured before it was presented to the public. We have passed our greatest expectations and I am safe in making the assertion that the sizer is well within the commercial limits of 90 per cent of the industries requiring closely sized material. There are, of course, special cases in this field as in others, that will require especially designed machines to successfully meet their requirements, but the fundamental principle of this machine is such that there is no question but that its development will eventually reach a point far beyond anything ever considered by the advocates of close sizing.

Comparing the work of this sizer to that of a screen, we find, to begin with, that the principle of the sizer is as far removed from that of a screen as is the principle of a classifier from that of a screen, but in the opposite direction. In other words, a screen is intermediate between the two principles. A classifier grades according to weight regardless of size, volume or bulk of particle. A screen grades according to the diameter of a particle and takes cognizance of two dimensions only. The McKesson sizer grades according to the bulk or volume of particle, regardless of its shape, and is affected by specific gravity only to a very limited degree, and that directly opposite to the effect of specific gravity in a classifier. This will be shown by the chemical analysis of the screen oversize and undersize in the products tested.

It is obvious that if a product, graded according to bulk or volume of particles, be tested on screens, there will naturally be a wide range between the two extremes in this product. That is, the difference, between a scale and a cube as measured by a screen which takes cognizance of two dimensions only will cover a relatively wide range, while in reality the volume of the particles may be identical.

[1] See October issue of this journal, p. 553.

The reverse is true of a screened product, for, while the product may be between screens of only a few points difference, the ratio of volume of largest to smallest particle may be several times the ratio of the cubes of the apertures of the screens limiting the product.

Quoting Dietz & Keedy, METALLURGICAL AND CHEMICAL ENGINEERING, November issue, page 578: "In the practical application of close and accurate screening we have adopted as a standard a series of thirteen screens between 20 and 200 mesh conforming to a harmonic series, thus producing fourteen products including oversize and fines. The average volumetric ratio is 2:1, while the efficiency, when measured by the screen actually limiting the grades, averages better than 90 per cent, the only drop under 90 per cent being in the —150 + 200 mesh grade, which, with the screen area usually exposed, varies between 80 per cent and 85 per cent. The series, as developed by us and used as a standard, is the following, in which screen cloths have been selected of apertures conforming to the law of cubes."

Mesh.	Apertures, Inches.	Ratio of Volumes of Largest to Smallest Particles.
20	.0410	1.722:1
24	.0342	1.722:1
28	.0282	1.783:1
34	.0229	1.867:1
42	.0183	1.959:1
50	.0145	2.012:1
62	.0116	1.953:1
74	.0089	2.214:1
86	.0068	2.242:1
109	.0054	1.996:1
125	.0041	2.284:1
150	.0032	2.103:1
200	.0026	1.865:1
Average		2.000:1

To show the fallacy of this system let us take the product between the screens 20 and 24 mesh, and assume a particle in the shape of a flat scale that will just correspond to the square of the aperture of the 24-mesh screen, but having a third dimension or thickness of, say, 0.0026 in., which is the aperture of the 200-mesh screen. It is obvious that this particle is bound to report between the screens 20 and 24 mesh, and will represent the smallest particle in that product with a volume of 0.000003041064 cu. in. Now take a cube that will correspond to the cube of the aperture opening of the 20-mesh screen, which is 0.00006892I cu. in. Here we have a volumetric ratio of largest to smallest particle of 22.66:1 instead of 1.722:1 as claimed. Take the next two screens, 24 and 28 mesh, and figuring on the same basis we get a ratio of 19.37:1 instead of 1.783:1. It is quite true that in some cases an ore is found that will have a fracture favorable to the application of the law of cubes, but unfortunately these cases are very much in the minority.

From the foregoing it is evident that in a product graded by screens it is a physical impossibility to get anything like the ratios claimed in all but a few rare cases where all the material will break in the form of cubes, and it is equally impossible to test by the use of screens with any degree of accuracy the efficiency of a machine which grades according to volume or bulk of particle.

Dietz and Keedy have taken the results of a test on the McKesson sizer, and have calculated the volume ratio of largest to smallest particles according to the aforementioned law of cubes. Taking, for example, their calculation for product No. 7 which apparently shows the largest ratio, their figures are as follows:

Product No.	Screen Mesh.	Efficiency per Cent.	Volume Ratio of Largest to Smallest Particle.
7	—50 + 82	90.63	6.20:1

A very careful test was made on this product to determine the actual, not theoretical, ratio of largest to smallest particle in this product. It was screened to give two products, viz., —50 + 66 which is the largest grade, and —74 + 82 which is the first grade in product No. 7. Equal volumes of these two grades were then taken by displacing 0.02 cc. of water, and the number of grains in each grade was accurately counted. There were 2200 particles in the coarse grade and 4165 particles in the fine grade. Now we have something definite to work upon and we find that there is an actual volume ratio of largest to smallest particle of 1.893:1 instead of 6.20:1 as calculated from the law of cubes. This accounts for the lapping of products so seriously objected to and shown graphically by Dietz and Keedy on page 578. A careful examination of the different products under a powerful microscope reveals the same condition throughout all the products.

Taking the case of product No. 15:

Screen.	Per Cent.	Efficiency.
—82	.....	
+97	6.25	
+116	18.75	
+139	43.75	
+172	25.00	68.75
—172	6.25	

We have here a product ranging from +97 to —172 which, when examined under the microscope reveals the scaly nature of the grades +97 and +116. In this case a scale corresponding to the square of the aperture of screen 97, which is 0.0058, and having a third dimension or thickness of one-seventh that amount, will give a particle +97 mesh which belongs in a 200-mesh product.

Now to take up the effect of specific gravity: From the analysis given herewith it is evident that whatever effect specific gravity has, it manifests itself in such a way as to remove all semblance of a doubt as to reversal of the accepted fundamental principles governing agitated material.

The attempt to show that the McKesson sizer is a dry classifier has led Dietz and Keedy into the error of predicting certain things regarding the nature of the products of this machine. They say, speaking of classified products, "In each grade it will be found that the lightest mineral reports to the greatest extent in the coarsest grade, while the heaviest mineral will be found to the greatest extent among the finer particles of the particular product under consideration. This same action is to be expected in a machine of the McKesson type."

In other words, they believe that if one of the McKesson products be subjected to a screen analysis the coarseat portion will contain the most gangue and light mineral, while the finest portion will carry the most mineral of high specific gravity, and the intermediate sizes will contain proportionate quantities of the minerals according to their specific gravities. They suggest a chemical analysis of the products from a mixed sulphide ore as being the best way to prove their contention.

With this object in view I have taken a product from such an ore and have had its several screen grades analyzed. The results are exactly opposite to what Dietz and Keedy predict, and show that in no sense is the product a classified material. In fact the analysis shows conclusively that the product is a sized material.

Screen Analysis on Size No. 8 of a Zinc-Iron Crude Ore.

Screen.	Per Cent.
—40	.....
+50	12.50
+66	75.00
+74	9.36
—74	3.12

According to Dietz and Keedy's anticipation that the product is classified material we should find a preponderance of gangue and lighter minerals in the coarse grade, and a similar excess of iron and zinc in the fine grade. If their contention is correct we ought to find the highest percentage of insoluble in the +50 grade, with diminishing percentages in the finer grades. Similarly, iron and zinc as the heavier minerals should appear in the different grades in increasing proportions. But such is not the case—indeed the facts are the reverse of Dietz and Keedy's assumptions, and disprove their idea that the McKesson machine is a dry classifier. Its different products are sized, and this has been proved many times by treating the most complex mixtures, the components of which were widely separated in specific gravities.

Denver, Col.

Chemical Analysis of Size No. 8 and Its Several Screen Products.

	Zinc, per Cent.	Iron, per Cent.	Insoluble, per Cent.
Size No. 8.....	29.20	12.00	15.56
Products of Size No. 8.			
+50	25.10	19.05	2.68
+66	32.90	13.30	8.90
+74	25.80	7.80	28.64
—74	14.80	5.40	48.08

Flameless or Surface Combustion.

A New Method of Using Gaseous Fuel for Heating and Steam-Raising Purposes.

By John B. C. Kershaw.

Sir Humphrey Davy, during his remarkable researches upon flame in the year 1816, discovered that under certain conditions a mixture of a combustible gas and air (in proper proportions) when forced into contact with a heated porous and refractory material or metal could *be burned without the production of any flame*, yet with the maintenance of a very high temperature. For various reasons this method of employing gas has made little progress, as regards its industrial application for heating or steam-raising purposes, and it is only to-day, nearly one hundred years after the first discovery of the method, that flameless or surface combustion has been developed into a practical and valuable industrial process.

Professor William A. Bone (F.R.S.), professor of applied chemistry in the Leeds (England) University, and Mr. C. D. McCourt have for some time past been engaged in the study of the theoretical and practical side of the subject, and have recently patented a process and apparatus, by means of which this method of combustion can be applied either for heating crucibles, furnaces or steam-boilers, with efficiencies that approach 95 per cent. of the actual heat value of the fuel gas used. The practical importance of this new method of using gaseous fuel for heating purposes *depends altogether upon this question of efficiency*. If, as stated by Bone and McCourt, thermal efficiencies of 90 per cent to 95 per cent can be attained in practice with all varieties of fuel-gas, whether for steam-raising or for furnace operations, the future of the new process would seem to be assured.

The one drawback attending the gasification of solid fuel in the past has been that the gas when employed for heating purposes (owing to its low radiant heat power) only yielded from 50 to 70 per cent of its heat to the object that required it, and that the direct thermal loss (apart altogether from the heat lost during gasification) amounted, therefore, to from 30 per cent to 50 per cent of the total heat value of the fuel.

The gasification of solid fuel and the employment of the gas for heating purposes have therefore only been adopted in the past under special conditions, when a very high temperature was required (as, for instance, in the glass and steel industries, where regenerative gas furnaces are employed), and it has been most unusual to find any steam-boiler or ordinary furnace heated by fuel gas, on account of the low thermal efficiencies obtained with this method of heating.

Bone and McCourt by rendering it possible to produce the greater portion of the heat as radiant heat at a very high temperature, and to attain heat transfer efficiencies of 95 per cent with gaseous fuel, have, however, brought this method of heating within the domain of practical application in the manufacturing industries. In the future one may expect to see a gradual extension of the plan of gasifying the solid fuel at the pit's mouth, recovering the ammonia and more valuable organic by-products of the destructive distillation of the coal in adjacent chemical works—and then selling the residual gas to manufacturers, either for power generation or for heating purposes. This method of utilizing fuel is undoubtedly the most scientific and the most promising, since it makes the best use of all the constituents of the fuel, involves the least waste and indirectly helps to solve the smoke problem.

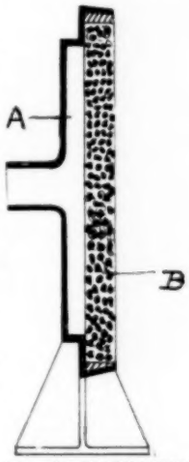


FIG. 1—DIAPHRAGM FOR SURFACE COMBUSTION.

The readers of this journal may therefore be assumed to have a special interest in it. In the following notes the writer will give some details of the apparatus used for the trial results obtained with the new method.

Fig. 1 shows in sectional elevation the simplest form of apparatus for applying surface combustion to heating purposes. *A* is the feed-mixing chamber through which the gas and air pass; *B* is the diaphragm at the surface of which combustion occurs. The amount of air used is in slight excess of that required for *complete* combustion of the gas.

The air and gas are forced into the mixing chamber, and

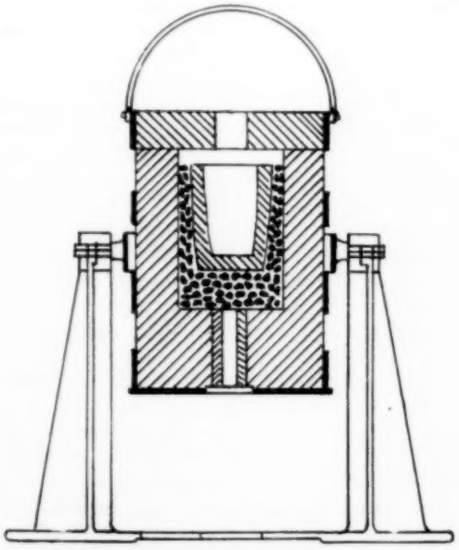


FIG. 2—APPLICATION OF SURFACE COMBUSTION TO A CRUCIBLE FURNACE.

therefore through the diaphragm, at a speed greater than that mixture; otherwise what is known as "backfiring" will occur. The prevention of "backfiring" and the choice of a suitable refractory material are in fact the two difficulties which have hindered the successful operation of this method of burning gas.

The gas-mixture burns without flame within a layer ⅛ in. (3 mm) in thickness, at the surface of the diaphragm, and maintains it in a high state of incandescence. The other parts of the apparatus are cold enough to touch with the naked hand, and the slightest variation in the pressure of the gas at which the explosion wave is propagated through the gas-supply causes an instant change in the degree of incandescence of the diaphragm, showing that there is no reserve of heat in the body of the same, and that the temperature attained by this method is under instant control. An incandescent surface of this type can be used for concentrating heavy liquids similar to solutions of sodium silicate and sulphuric acid by surface evaporation.

Fig. 2 shows the new method applied to the heating of a

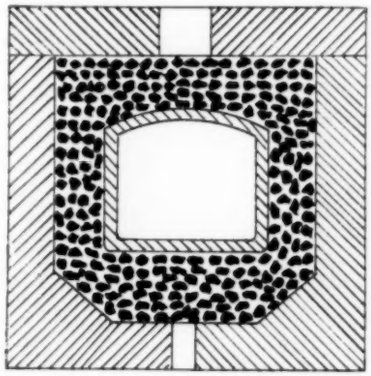


FIG. 3—APPLICATION OF SURFACE COMBUSTION TO A MUFFLE FURNACE.

crucible tipping furnace and Fig. 3 to a muffle furnace. In these cases a bed of well-graded pieces of the refactory material surrounds the crucible or muffle and the explosive gas-mixture is blown into this bed through the opening shown in the diagrams at the bottom of each furnace. As the gas-mixture impinges upon this incandescent bed, combustion is completed almost instantaneously, and the products of combustion as they rise through the upper portion of the bed rapidly impart their heat to the material composing it.

A charge of cast iron can be melted in one of these crucibles in ten minutes, starting with cold metal, and the only limit at present as regards temperature is that imposed by the melting or decomposing point of the refractory materials employed. Carborundum (silicon carbide), which has been widely used for the purposes of a furnace lining and is considered highly refractory, is decomposed quickly into CO_2 and SiO_2 in the

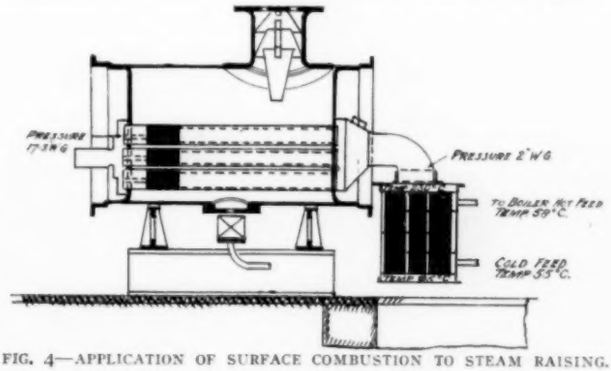


FIG. 4—APPLICATION OF SURFACE COMBUSTION TO STEAM RAISING.

interior of these furnaces, indicating that a temperature of 2000° C. has been approached.

With mixtures of coal gas and air, or water gas and air, in theoretical proportion, injected into the furnaces at a maximum speed, temperatures have been attained which no material hitherto used for furnace or muffle construction can long withstand. Gases of lower calorific value, such as coke-oven and blast-furnace waste gases, of course yield lower temperatures. No unburnt gas escapes by this method of combustion when once the correct proportions of gas and air have been properly adjusted, and, therefore, the maximum

temperature possible to the gas is always obtained unless, as already explained, an excess of air is purposely admitted to reduce the temperature.

Fig. 4 indicates how this principle of combustion can be applied to steam-raising purposes. The experimental boiler shown in diagrammatic longitudinal section in this illustration is provided with ten fire-tubes, each 36 in. in length and 3 in. in diameter. Each tube is packed for ⅞ of its length with the refractory material, broken and graded into pieces of suitable size.

At the front end of each tube is a fireclay plug provided with a central hole ¾ in. (18 mm) in diameter. This plug serves the double purpose of keeping the front end of the boiler cool and of providing a suitable aperture through which the explosive gaseous mixture can be injected at a speed very much higher than the speed of "backfiring."

Attached to the front plate of the boiler is a mixing-chamber of special design. The economizer at the rear end of the boiler is provided with nine tubes, each 3 in. in diameter and 12 in. in length, and packed as before with the refractory material, suitably graded.

Under working conditions this experimental boiler showed *complete combustion* of the gaseous mixture before it had traversed half the length of the fire-tubes. The central portion of the refractory core of each tube was maintained at a very high temperature; this was rapidly transferred to the water in the boiler by radiation.

As regards the rate of combustion, about 1000 cu. ft. of coal gas have been burned per hour, in this small boiler. The temperature of the exit gases is from 60° to 70° C. higher than that of the water in the boiler, but the economizer intercepts the larger portion of this heat, and reduces the temperature of the exit-gases to less than 100° C.

Sixty-five per cent of the total evaporation of the boiler occurs over the first foot length of the fire-tubes. This causes violent ebullition and rapid circulation, the latter being an important factor in obtaining rapid heat transmission.

In an ordinary Lancashire boiler the rate of heat transmission is equivalent to the evaporation of 6 to 7 lb. of water per square foot of heating surface per hour; in the locomotive boiler this is increased to 10 lb.; whereas in the boiler just described it rises to 20 lb.

The following are figures obtained during a test run of the boiler:

The amount of coal-gas fed into the boiler, expressed as dry gas at 0° C. and 760 mm, was 936 cu. ft. per hour. The net calorific value of the gas was 562 b.t.u. per cubic foot—the total heat supplied to the boiler in terms of the net calorific value of the gas was 559,800 b.t.u. per hour.

The actual amount of water evaporated was 450.3 lb. per hour, which in terms of water evaporated from and at 100° C. would be 550 lb. per hour. The actual heat transmitted to the water was $450.3 \times 1172 = 527{,}800$ b.t.u. per hour. The ratio between the heat transmitted to the water and heat supplied was, therefore, $\frac{527{,}800}{559{,}800} = 0.943$.

The pressure under which the gas-mixture is blown into the boiler is equal to 17.3 in. (water gage), while the exit gases escape to the economizer at a pressure of 2 in. The resistance to the flow of gas, offered by the fire-clay plugs and the packing of refractory material in the fire-tubes, explains this steep pressure gradient. According to the inventors of this system, the actual power expended in drawing or forcing the gases through the boiler and economizer is only 4 per cent of the total power generated. This reduces the net efficiency of the method to 90 per cent, which is still considerably higher than that obtained with coal or gas-fired boilers.

Another advantage of this system is the rapidity with which steam can be raised. Starting with the ten-tube boiler all cold it is possible to obtain steam at 150 lb. per square inch pressure in about twenty minutes, and to continue the high rate of evaporation indicated—viz., 20 lb. of water from and at 100° C. per square foot of heating surface per hour—so long as the gas supply is properly maintained.

If, in order to meet an "emergency" demand for steam, there is occasion to force the boiler so as to obtain an evaporation of say 30 lb. of water (from and at 100° C.) per square foot of heating surface per hour, all that is necessary is to force more of the gaseous mixture through the boiler tubes, and to sacrifice 5 per cent in efficiency.

A final advantage of the system, as applied to steam-raising purposes, is that in a large boiler the tubes could be arranged in groups, and that the gas mixture could be cut off, or put on to, any group of tubes, according as the demand for steam was diminished or increased.

This elasticity of the system is claimed by the inventors as of great value and advantage in those manufacturing industries where a widely fluctuating load has to be carried by the boilers. A large boiler fired on this system is now being erected at Leeds by the Radiant Heat Syndicate, which has taken over the industrial development of the Bone and McCourt patents, and the test results obtained with this boiler will be awaited with great interest by all power engineers. As already stated, this novel method of combustion seems to open up a new era of progress in the application of gaseous fuel to manufacturing and industrial purposes.

Direct Smelting of Argentiferous Copper Ore at Rio Blanco, Chile.

By Ignacio Diaz Ossa.

The ore deposit known as Cristobal Colon occurs almost on the summit of the chain of mountains forming the left side of the Rio Blanco valley, and is situated 5 km from a station on the Transandino Railroad. The deposit consists of a narrow vein of high-grade copper ore, carrying high silver values, crossing three thick beds of mineralized volcanic turf.

The vein was worked by a day-level until the first bed was cut; a shaft was then sunk at the junction of the vein with the bed, when two other beds were discovered. These have been attacked along the face. During the year the output amounted to 1800 tons.

The ore is raised by winches to the level of the adit, which is about 25 m above the lowest bed. Near the mouth of the adit the ore is trammed along the edge of a precipice for about 200 m to the upper station of a ropeway connecting the mine and smelter, the latter being erected on the banks of the river. The ropeway is in two stages, and usually remains idle from 1 to 3 p. m. daily, owing to the high winds which prevail at that time.

The beds, the total thickness of which amounts to about 15 m, contain on an average 5 per cent copper. Rapid hand sorting raises this percentage to 8, and the value of the silver to about 75 pesos per ton. (One peso = $0.23 gold.)

Smelting Operations.

The blast furnace, which is of the cylindrical type, is defective and not built on modern lines. Its daily capacity is 20 tons. The dimensions are as follows: Height from charging floor to tuyeres, 2.17 m; diameter of cylinder at charging floor and tuyeres, 1.10 m; number of tuyeres, 12; diameter of tuyeres, 0.17 m; length of water jacket, 1.53 m; upper part, covered with brick, 0.87 m; height from tuyeres to crucible, 0.23 m. The blast is provided by a Root blower operated by two Pelton wheels producing 25 hp. A short channel conducts water from the Blanco to the wheel house.

On taking charge of the smelter I did away with the old heavy forehearth on rails, and made an inner crucible 1.40 m deep, with a diameter at the floor of 0.58 m; and another 1.10 m below the level of the tuyeres. I also dispensed with the

lip for the discharge of slag, making an arrangement with firebrick which gave much better results.

The following figures show the percentage composition of the ore: Silica, 45 to 50 per cent; lime, 8 to 12; iron, 7 to 11; sulphur, 4 to 6; alumina, 7 to 10; lead, 1.5 to 3; copper, 8.9 to 12; arsenic, 0.1; silver, 700 to 800 grams per ton.

As the only flux obtainable was a dolomite limestone containing 8 to 10 per cent heavy spar, old reverberatory slags had to be used as well. Analyses showed them to have the following percentage composition: Silica, 45 per cent; iron, 31; lime, 12; alumina, 8; copper, 2.5; silver, 150 grams per ton.

The ore, flux and slag were mixed in such proportions as to form a bisilicate of lime. The resulting slag analyzed as follows: Silica, 50 to 52 per cent; lime, 26 to 30; iron, 10 to 12; alumina, 6 to 9; copper, 0.15; silver and lead, traces only. The slag ran well at a temperature exceeding 1250 deg. C. Both iron and sulphur oxidized easily, the final products being matte containing 75 per cent copper, and copper in bars. About 25 per cent of the copper was obtained in bars of 90 per cent metal. The copper matte contained from 7 to 9 kg of silver per ton.

In ten days we smelted 54,530 kg of ore carrying 12 per cent copper and having silver to the value of 60 pesos per ton; 12,910 kg of ore of 7 per cent copper and 40 pesos silver per ton; 38,200 kg of ore and slag of 2.5 per cent copper and up to 10 pesos silver per ton; 176,000 kg of flux, and 30,200 kg of coke. The total costs amounted to 8576.8 pesos, and the value of the product was 14,718 pesos, giving a profit of 6141.2 pesos. With proper fluxes, which are now being accumulated, the profit should be about double the above.

Cooling in the crucible was reduced by increasing the blast. In order to prevent as much as possible firing at the level of the charging floor, and consequently black tuyeres, charges of 450 kg were used, the coke and charge being moistened with water. The coarse ore and fuel were charged around the circumference of the furnace, and the fine material in the center. This prevented small pieces passing down the vertical sides of the furnace without being properly calcined.

No doubt many ores can be smelted to a matte containing a high percentage of copper, and even to metallic copper, provided pyrites is not added to the furnace charges, but only calcareous or ferruginous oxidized fluxes, such as basic slags, hematite, etc. Loss in the slag is generally said to be the result of producing mattes of high copper content, but in the Rio Blanco smelter, with a high crucible, the slag never rises above 0.16 per cent copper, and the average is only 0.12 per cent.

Santiago de Chile, S. A., Casilla 962.

Tests on Monel Metal Castings.

The Bayonne Casting Company, which has succeeded in making castings of monel metal up to 25,000 lb. in weight, has made some interesting tests of such castings under U. S. Navy inspection.

This new "natural alloy" bids fair to displace steel and bronze in a large number of cases as it has been found to be as strong as steel and is less corrodible than bronze, while it takes a finish like pure nickel. Below are the results of the tests and the U. S. Navy specifications for monel metal castings.

Bureau of Steam Engineering, United States Navy specifications for monel metal castings are as follows:

Yield point (minimum).....	32,500 lb. per sq. in.
Tensile strength (minimum)...	65,000 lb. per sq. in.
Elongation in 2 in. (minimum)	25%

An average of tests of monel metal from 25 heats made under U. S. Navy inspection gave:

Yield point	35,197 lb. per sq. in.
Tensile strength	72,843 lb. per sq. in.
Elongation in 2 in.....	28.2%
Reduction of area.....	35.6%

The Electric Reduction of Iron Ores.

With Special Reference to Results Obtained in the Electro-Metals Furnace at Trollhättan, Sweden, and in the Noble Furnace at Heroult, California.

BY OTTO FRICK.

The publication of Mr. J. A. Leffler's report* on the Trollhättan furnace, presented at the annual meeting of "Järnkontoret" in Stockholm on May 31, 1911, offers an opportunity to discuss the possibilities of the electric reduction of iron ores.

The writer has not had an opportunity to personally inspect the Trollhättan furnace, but Mr. Leffler's report gives a very detailed account, both of the construction and of the working of this furnace. The information with regard to the Noble furnace has mainly been collected during a visit at Heroult, Shasta County, Cal., last year.

Type and General Design of Furnaces.

The Trollhättan furnace and the Noble furnace belong to the same type, and are on most essential points alike. This type has been developed from experiments carried out on different lines and without any mutual understanding or communication.

Both furnaces consist of a wide melting chamber superposed by a shaft of smaller diameter so as to allow the introduction of electrodes through the roof above the crucible—a type of furnace which has been proposed by Keller already in 1901.

The main advantage of this type of furnace is the possibility of using a high shaft and introducing the electrodes in the lower part of the furnace without exposing them to too great a pressure from the charge and the possibility of producing a hot zone on the outside of the charge column at some distance from the brick work.

The shape of the shaft is in both cases of a similar form as used in an ordinary blast furnace, that is to say conical towards the melting zone as well as towards the furnace top. This shape is preferable to a cylindrical shaft, as it allows to increase the volume of the shaft without interfering with the dimensions of the melting chamber, which in an ordinary blast furnace has to be narrower than the shaft to allow the blast to reach the center, and in the case of the electric furnace has to be wider than the shaft to allow the introduction of the electrodes through the roof of the said chamber.

In the first furnace of the Electro-Metal type, built at Domnarfvet, Sweden, the shaft of the furnace was supported on iron pillars. In the Trollhättan furnace the shaft is hung on strong beams, in a similar manner as used in the Noble furnace, erected one year before the Trollhättan furnace, which arrangement is preferable to the previous one as it secures free space all around the melting crucible.

It has been claimed that this shape in the case of an electric furnace should offer special advantages not present in an ordinary blast furnace, in so far as it should prevent the charge in the melting chamber from being pressed against the roof, an effect pretended to occur with a shaft of cylindrical shape. This is, however, not the case, as the shape of the shaft has no influence on the behavior of the charge in the melting chamber.

Whether the lower part of the shaft is conical or cylindrical, there will be the same pressure at the neck of the melting chamber, and if the temperature of the charge at this point is below the one at which the material gets pasty, the charge will in both cases take a conical shape in the melting chamber, the surface of which will form an angle of about 35° to the horizontal. Only if the temperature at the neck were high enough to make the charge enter into the melting chamber in a semi-fluid state there would be a danger of the charge spreading against the roof.

As it is under no circumstances desirable, and, in fact, next to impossible to work the furnace in such a manner, no such danger needs to be feared. In later runs of the Noble furnace

*See this journal, July issue, p. 368, September, p. 459, October, p. 505.

the shaft has obtained a cylindrical shape without altering the behavior of the charge in this respect.

Size and Power Consumption.	Trollhättan Furnace.	Noble Furnace.
Maximum power consumption	2,500 hp 1,840 kw	2,040 hp 1,500 kw
Total height of furnace	13,700 mm	8,300 mm
Outside diameter of crucible	5,650 mm	3,960 mm
Outside diameter of shaft	3,200 mm	2,790 mm
Volume of shaft	25.5 cu. m	7.9 cu. m
Radiation surface	189 sq. m	105.5 sq. m

From these figures it will be seen that the dimensions of the Trollhättan furnace are considerably bigger than that of the Noble furnace.

The diameter of the crucible is mainly governed by the size of the electrodes which have to be introduced through the roof. By the use of graphite electrodes, and also by using the greatest possible number of electrodes, a saving in the diameter of the crucible is obtained.

The size of the shaft in the Trollhättan furnace has been calculated for an average ouput of 23 tons of pig in 24 hours, and on the assumption that the proportion:

$$\frac{\text{volume of charge in 24 hours}}{\text{volume of shaft}} = 1.55.$$

This figure is evidently based upon the average practice of the Swedish charcoal blast furnaces, in which it varies between 1.21 and 1.91.

The author of the Trollhättan report expresses the opinion that the correctness of this calculation has been substantiated by the results obtained, but it appears as if these results in reality show that the shaft is unnecessarily big.

In a paper read by the present writer at Järnkontoret in Stockholm in May, 1905, it was shown that the electric reduction furnace theoretically only needs to be 1/12 of the size of that of an ordinary blast furnace.

The main object of a shaft on any furnace is to recuperate the heat of the gases from the melting zone, and if possible to give the carbonic oxide an opportunity to reduce part of the ore.

In an ordinary blast furnace the weight of the gases exceeds the weight of the ore, limestone, and coke or charcoal by 30 per cent to 50 per cent.

In an electric reduction furnace the gases evolved amount only to about 40 per cent of the weight of the charge.

By the use of a circulating gas introduced in the lower part of the furnace, it is, however, possible to increase the total weight of the gases passing through the shaft so much as to make it exceed the weight of the charge in a similar proportion as in an ordinary blast furnace, but even then a considerable difference between the blast furnace and the electric furnace remains in so far as the temperature of the gas entering the lower part of the shaft cannot be as high in the electric furnace as in the blast furnace unless the circulating gas is heated outside the furnace by the use of hot blast stoves.

In the Trollhättan furnace such an arrangement has not been provided. The heat, carried by the circulating gas to the shaft, has thus to be absorbed from the melting zone, and is necessarily restricted by the surface and the temperature of the charge through which the gas is passed.

The result of the Trollhättan furnace demonstrates to full evidence that this surface and this temperature just mentioned are not sufficient to allow the gases to reach more than 500° C.

From the records collected the highest temperature ever observed at the bottom of the shaft has been 682° C, and the lowest has been 241° C, while the average temperature at this point has been 510° C.

The corresponding temperature in a blast furnace is considerably higher and probably a great deal above 1000° C. The total heat stored in gases at 1000° C. is nearly 2.5 times bigger than that at 500° C. It is, therefore, evident that with the same amount of gas per ton of pig the volume of the shaft of an electric reduction furnace can be made 2.5 times smaller than the corresponding shaft of an ordinary blast furnace with an equally satisfactory transfer of the heat from the gases of the charge. Whether it is consistent with economical working to use so much circulating gas will be discussed later on.

The volume of the shaft of the Trollhättan furnace is 25.55 cu. m.

The corresponding volume in the case of the Noble furnace has been only 7.9 cu. m., and this has proved to be more than ample. In later runs the volume of the shaft has been further reduced by being made cylindrical, apparently without interfering with the economy of the process.

The temperature of the gases leaving the shaft of the Trollhättan furnace has varied between 205° C. and 29° C. and has on an average been 76° C.

These figures compare well with the temperature generally held by the gases from a blast furnace, which as a rule is above 200° C. It has, however, to be kept in mind that the total heat contained in the gases in a blast furnace, say at 1100° C., exceeds the heat which can be absorbed by the charge, whereas the heat of the produced gases in an electric furnace is very much less than that required by the charge.

By varying the amount of circulating gas in the electric furnace, the total heat brought into the shaft by the gases can be made to equal or to exceed or to become less than the heat which can be absorbed by the charge.

Electric Arrangements.

The Trollhättan furnace is supplied by two single-phase transformers, each of 1100 kilovolt amperes, on the primary side connected to a three-phase system of 10,000 volts, 25 periods, according to the Scott method of transformation from a three-phase to a two-phase system. From the transformers a two-phase current is delivered to the four electrodes of the furnace.

The Noble furnace has three single-phase transformers each of 750 kilovolt amperes connected to a three-phase system of 2200 volts and 60 periods.

The low-tension current from the transformers is supplied to the six graphite electrodes.

In both furnaces the regulation of the current is effected by a variation of the number of turns of the primary coils of the transformer. The priority of this device belongs to the Noble furnace for which it was designed by the General Electric Company in the early part of 1909.

The arrangement is well known for other purposes and it is only natural that it has been adopted for electric reduction furnaces as it affords a simple and effective means of maintaining a steady power consumption, and is the only possible way if the furnace has not an individual dynamo but is connected to a great power distribution system.

At the Noble furnace it has also been found useful to use the regulating coils alone, and thus to increase the secondary voltage for the purpose of starting the furnace. It was found, for instance, that with cold charcoal and the normal voltage of 70 volts no current would pass through the charge, and only by raising the voltage to about 100 volts in the above explained manner it became possible to make the current start.

This difficulty will evidently not arise if the charge is preheated to a sufficiently high temperature by blowing air into the crucible.

Electrodes and Method of Heating.

The Trollhättan furnace and Noble furnace differ essentially with regard to the method of heating and to electrodes.

The former furnace has four ordinary carbon electrodes of 660 × 660 mm, which enter into the charge only slightly, as it is endeavored to keep them "free burning"; that is to say, to produce a small space between the electrodes and the charge across which electric arcs are formed.

The latter furnace has six graphite electrodes of 8½ in. diameter—216 mm—which enter into the charge as far as

possible so as to prevent them from becoming "free burning" and so as to insure best possible contact between the electrodes and shaft.

In the report on the Trollhättan furnace it is said that "free burning" is an essential feature, as otherwise the voltage becomes unnaturally low and the dimensions of the electrodes and leads must become excessive and that finally too great variations of the power consumption would result.

It is noteworthy that several of the electrodes of the Trollhättan furnace must have entered into the charge quite considerably, as demonstrated by the sketches given in the report. The electrode III, for instance, appears to have entered into the charge with about 90 cm. This electrode shows also the smallest net electrode consumption per ton of steel.

In using "free burning" electrodes the main resistance in the path of the current between the electrodes, and between the electrodes and the bottom, if this is made to form one pole of the circuit, is concentrated in the arcs near the electrodes, so that nearly the whole of the electric energy is commuted into heat at this point. To convey this heat from the electrodes to the different parts of the charge it is necessary that the temperature near the electrodes be essentially higher than that required for the reduction of the ore and for the melting of the pig and the slag.

A further inconvenience of "free burning" electrodes is that the hottest zone is generated near the surface of the charge and thereby causes difficulty in protecting the roof from the heat. This difficulty has, however, in the Trollhättan furnace been proved to be overcome by introducing a suitable amount of cooling gas around the electrodes.

The introduction of the electrodes into the charge to such an extent as to prevent their "free burning" reduces the resistance at the contact between the charge and the electrode, and thereby the amount of heat produced by the passage of the current at this point and increases the heat generated by the current in passing through the charge. Less heat has thus to be transferred from the zone around the electrodes and a lower temperature would thus be required. As the further end of the electrode is at a considerable distance from the surface of the charge, there is no danger of over-heating the roof and no gas circulating for the purpose of cooling the roof will be necessary.

The difficulty which this arrangement causes is undoubtedly more serious in the case of ordinary carbon electrodes than in the case of graphite electrodes.

There is no doubt that it is not the extra cost of cables and electrodes which would prevent the use of the method applied in the Noble furnace.

The essential difficulty is really to be sought in obtaining electrodes of sufficient length and strength.

The great size of ordinary carbon electrodes makes it difficult to manufacture them in great length or to provide them with a reliable joining device.

Also the jointing of graphite electrodes offers certain difficulties. At Heroult the one end of the electrode is shaped into a conical screw fitting into the upper part of another piece of electrode.

The cutting of this screw joint is very easy on account of the softness of the material, and the joint makes a perfect electrical connection, but the strength of the electrode is considerably reduced and has proved not to withstand the pressure exercised by the charge.

In the Noble furnace the pressure of the charge on the electrodes is very nearly equal to their breaking strength, so that the additional force arising from a sudden descent of the charge easily causes the breaking of the electrodes at the joint. By proper design the electrodes can, however, be made strong enough.

In the case of the Noble furnace the patrice part of the joint was essentially weaker than the matrice part, and only by giving them equal strength, the total strength of the electrodes can be increased by as much as nearly one hundred per cent.

The strain on the electrode, caused by the charge, also depends on the inclination given to the electrodes. This inclination is 35° to the vertical, and by reducing the same to 20° the strain on the electrode can be approximately reduced by about 30 per cent.

With graphite electrodes of bigger diameter than 8½ in. still less trouble from this source needs to be experienced.

The method of introducing the electrodes into the charge and thus working the furnace as pure Resistance Furnace requires a bigger current and a lower voltage than in a furnace with "free burning" electrodes.

With increasing amperes it is of great importance that provisions be made to avoid inductive losses along the path of the current. In this respect, the care taken at the Trollhättan furnace does not seem to equal that at the Noble furnace, in so far as the inductive voltage drop per 1000 amperes in the former furnace appears to be 2.5 volts with a periodicity of 25 as against 2.8 volts in the Noble furnace with 60 periods, so that the inductance in the former case is more than double that in the Noble furnace.

Consumption of Electrodes.

The consumption of electrodes in an electric Reduction Furnace is of no small influence on the economy of the process.

The average consumption during the run of the Trollhättan furnace has amounted to 10.28 kg, costing about 75 cents per ton of pig produced. The actual consumption from wear has, however, only been 5.27 kg, so that a waste of nearly 50 per cent has been caused by no arrangement for jointing the electrodes having been provided.

It is, however, to be noted that the total electrode consumption during the four periods to which the report refers has dropped from 11.24 to 7.45 kg. The cause of this reduction in the electrode consumption is probably to a large extent due to the high average content of CO_2 in gases during the first period, but also other causes may have been of influence.

With reference to the consumption of graphite electrodes due to the wear and tear, no reliable figures are available as the breakage has been very big from causes mentioned above.

From the inspection of such broken electrodes which had been in use for some days the actual wear does not seem to have been appreciable.

As there is no arc between the electrodes and the charge, no particles from the electrodes will be carried away by arcing and the erosion of the electrodes can only be caused by the oxygen in the charge or in the circulating gas.

Taking into consideration the much smaller surfaces of contact as against an ordinary carbon electrode, and also the greater resistivity against oxidation, it is not unlikely to be expected that the actual wear of graphite electrodes will be less than that of ordinary carbon electrodes.

At the present price difference, and disregarding the advantages of graphite electrodes in other respects, the wear of these electrodes should only be 40 per cent of that of carbon electrodes to make this item of cost equal for both kinds of electrodes.

Charcoal Consumption.

The average coal consumption of the Trollhättan furnace during the whole run to which the report refers has amounted to 418 kg of charcoal per ton of pig besides the carbon derived from the electrodes and from the coke and electrode pieces charged. As this figure appears to be rather high it has been thought of interest to investigate this question more closely.

For this purpose it has been necessary to figure out the average analysis for all the materials which have been charged into, respectively delivered from the furnace.

In Appendix I these analyses are recorded. In this appendix are also noted the total amounts of material per ton of pig.

Appendix II contains a complete investigation of the different

purposes for which the carbon has been used during the different periods of operation, with the view of finding out the causes of the differences in coal consumption during different periods.

At first the total weight of gases per ton of pig are calculated from the average gas analyses and the total weight of C charged into the furnace. The total weight of gases produced per one ton of pig varies between 816 kg and 995 kg, and averages 855 kg.

The next calculation aims to find out the value of:

$$\frac{\text{Volume CO}}{\text{Volume CO}_2} = \frac{\text{Weight of C burnt to CO}}{\text{Weight of C burnt to CO}_2} = K$$

To arrive at this value the CO and CO_2 contained in the charcoal and limestone have been subtracted from the total amounts contained in the gases.

The value of K which may serve as an indication for the economy of the process varies between 2.57 and 5.58 and averages 3.71. This figure does not differ very much from the figure obtained in an ordinary blast furnace.

Next come the calculations made to find out the amount of oxygen to be reduced from water and ore, and of oxygen introduced into the furnace by air.

The weight of oxygen from water is found with the aid of the hydrogen in the gases reduced by the amount of hydrogen in the charcoal.

The weight of oxygen from the air is calculated on the assumption that the nitrogen in the gases is derived from air introduced into the furnace or into the piping connected with the circulating fan.

The amount of oxygen from ore is calculated on the basis of the analysis of the pig, and the average analysis of the oxygen connected to Fe in the ore, so arrived at, varies from 434 and 460 kg, and averages 445.7 kg.

These figures are very closely in accordance with those previously found from the analysis of the gases.

The amount of C necessary for the reduction of the oxygen is calculated from the formula:

$$C = 0.75\frac{K+1}{K+2} \times O$$

and should vary between 0.586 and 0.651 kg C for one kg O, the average value for the whole run being 0.619. The calculated weight of C varies between 285.58 and 332.22 kg per ton of pig and differs only by a few per cent from the actual weight of C supplied.

The result is most instructive, as it furnishes full information about the manner in which the carbon has been consumed, and thereby indicates by what means it would be possible to reduce the coal consumption.

From a study of the figures it will be seen that quite an appreciable part of the carbon, 10 per cent to 15 per cent, is consumed in the reduction of the water introduced by the circulating gas.

This part of the carbon is saved in a furnace without any gas circulation and can also be saved in the case of a furnace with circulating gas if provisions are made to take the gas from the furnace at a level below that at which the water is evaporated and by preventing the gas from absorbing any moisture in cleaning the same from the dust.

The only way to diminish the amount of carbon used for the reduction of the ore is by striving at a lower value of "K." In this respect it is noteworthy that the best value of "K" obtained during the second period coincides with the smallest amount of limestone and that during the third period "K" reaches the high value of 5.58, indicating a rather uneconomical working at the same time as the weight of limestone per ton of pig is highest or 336 kg per ton.

This should indicate that the coal consumption in an electric furnace will be reduced by using burnt lime instead of limestone, and as the burning of the limestone can economically be done by using part of the escaping gases for the purpose it thus will save current as well. It should always be made a rule to burn the limestone outside of an electric furnace.

During the period 2 practically all of the Fe_2O_3 has been reduced to FeO by CO. It is scarcely to be expected that any FeO can be reduced by CO under the conditions at present prevailing in a furnace of the Trollhättan type, and it is therefore not probable that the amount of charcoal for the reduction of the ore can be brought down further than during the said period 2 unless some radical alterations of the process are introduced.

Power Consumption.

The average actual power consumption for one ton of pig has been 2391 kw-hours. This corresponds to 3.66 tons per kw-year or 2.692 tons per hp-year if the power were fully utilized during the 8760 hours of the year.

It has become the general practice to express the output of a furnace in tons per kw-year or in tons per hp-year, but it ought to be pointed out that these figures may in fact be rather deceptive, as they do not take into consideration stoppages and loss in output by the furnace not always working at full capacity.

Only in very rare cases electric reduction furnaces will obtain the electric current on the basis of a meter tariff, thus only paying for the actual amount of electricity consumed. As a rule the electrical energy will have to be paid for at a fixed price per year for a certain amount of maximum available power.

In Table 1 the production per kw-year, according to these two different modes of expression, is given for the different periods of the run. From this it will be seen that the tons produced per hp-year to be paid for have varied between 1.315 and 2.7, with an average of 1.88 tons, whereas the production per hp-year, according to the report, has varied between 2.43 and 3.0, with an average of 2.69 tons per hp-year.

It is evident that it is most misleading to people not thoroughly conversant with the question to state the production per hp-year actually consumed, instead of hp-year to be paid for, and it may correctly be suggested that if the production is stated in the former figure it should be accompanied by a statement of the load factor.

As it has been deemed of interest to investigate how the electricity has been spent, an approximate calculation of the power consumption during the different periods has been made in the Appendix III.

The heat required for the reduction has been calculated from the analyses of the pig produced and with due regard to the heat developed by the combination of the oxygen in the ore with the carbon.

The heat necessary for the reduction of the water carried into the furnace by the circulating gas has been found in a similar way.

The heat absorbed by the calcination of the limestone can easily be found from the analyses of the limestone and the weight thereof charged.

In the calculation of the heat necessary for the evaporation of water no consideration has been taken of the water which escapes from the furnace, the amount of which could only be accurately ascertained by knowing exactly the amount of circulating gas. As it seems probable that this has been three to four times bigger than the amount of escaping gas the evaporation of the water in the escaping gases will require one-third to one-fourth of the heat calculated for the evaporation of the water in the circulating gas and would then amount to 10,000 to 15,000 calories per ton of pig.

This undervaluation of the heat necessary for the evaporation has in some way been counterbalanced in calculating the heat carried away by the gases.

As no figures with regard to the temperature of the pig and slag are given, the heat carried away by them can only be approximated and it has been assumed that each kilogram of

TABLE I.—POWER CONSUMPTION.

Period.	1.	2.	3.	4.	Total.
Number of charges	1–10	11	12–17	18–29	1–29
Tons of pig produced	1,154.5	145.5	248.0	332.1	1,882.5
Duration of mn, in hours	2,115.6	189.5	660.25	528.75	3,501.9
Duration of mn, in year	0.242	0.0216	0.0755	0.0603	0.4
Total kw-hours used	2,651,029	312,601	650,480	877,706	4,500,596
Total kw-year used	303	35.7	74.3	100.8	514.8
Total kw-year, to be paid for	445	39.7	139	111	737.0
Load factors	68.1%	89.7%	53.6%	90.3%	60.9%
Kw-hours per ton of pig	2,296	2,149	2,623	2,643	2,391
Kw-year per ton of pig used	0.262	0.246	0.300	0.302	0.273
Kw-year per ton of pig, to be paid	0.335	0.273	0.561	0.335	0.392
Tons of pig per kw-year used	3.82	4.08	3.34	3.31	3.66
Tons of pig per hp year used	2.81	3.0	2.46	2.43	2.69
Tons of pig per kw-year, to be paid	2.98	3.66	1.78	2.98	2.55
Tons of pig per hp-year, to be paid	2.19	2.70	1.315	2.19	1.87

pig has contained 283 calories and each kilogram of slag 500 calories, figures which should not be far from accurate.

In adding up the total power consumption the same has been expressed in kw-hours.

The total kw-hours theoretically required varies according to this calculation between 1536.3 and 2153.6 kw-hours per ton and averages 1721.4 kw-hours per ton. The actual energy consumption has varied between 2149 and 2710 and has averaged 2391 kw-hours per ton.

To this consumption of electric energy should be added the heat generated by air entering into the furnace with the circulating gas. The amount of this heat has varied between 13.95 and 35.5 kw-hours per ton, making the total actual heat consumption to vary between 2184.5 and 2735.8 and to average 2416.3 kw-hours per ton.

The difference between this actual heat consumption and the theoretically calculated energy consumption should correspond to heat losses from radiation from furnace walls, electrodes, leads and transformers, including the heat carried away by the cooling water used at the neck of the furnace and round the electrodes.

Calculations of this kind must always be more or less approximate and cannot be expected to give an absolutely correct image of what is actually taking place, but the information with regard to the distribution of the heat on different purposes will nevertheless be sufficiently accurate.

On studying the different items on which the heat has been expended it is found that a reduction of the energy consumption can be obtained by the same means which will secure a reduction of the coal consumed; that is, by preventing water from being introduced into the crucible with the circulating gas and by using burnt lime instead of limestone in the charge.

A further economy can be secured by drying the charcoal outside the furnace to get rid of the water.

By these precautions the theoretical energy consumption can be reduced by 136 to 232.8, or on an average 208 kw-hours per ton. The actual economy would vary between 195 and 358 and average 294 kw-hours per ton, the economy so secured would amount to from 9.1 to 15.2 and average 12.3 per cent. of the total power consumption.

A further saving will ensue from the use of burnt lime instead of limestone in so far as a lower value of K can be reached.

The average calculated efficiency of the furnace has been 71 per cent. This figure is hardly satisfactory for a furnace of this size, but is to a great extent due to the average low load factor.

In Appendix III it has been attempted to arrive at a distribution of the losses as close as possible corresponding with the actual conditions. The losses in the iron of the transformers and in the copper can be fairly accurately calculated from the information given in the report about the efficiency of the transformer and the size of the leads.

The heat absorbed by the cooling water has been measured at several opportunities and found to vary between 5.3 per cent and 8.2 per cent and average 6.56 per cent of the load.

The report also contains observations of the surface temperature of the furnace, which observations enable one to calculate the radiation losses to be about 200 kw, to which figure has to be added the radiation from the upper part of the electrodes with about 20 kw.

Comparing the losses calculated in this manner with the figures arrived at by the subtraction of the theoretical power consumption from the actual, it will be found that with the exception of the periods 2 and 3 the agreement is very fair.

Taken over the whole run, the two figures are exactly alike, which will indicate that the calculated distribution of losses is not far from being correct.

An appreciable reduction of the losses can only be gained by diminishing the dimensions of the furnace and by suitable precautions to prevent too much heat being carried away by the cooling of the electrodes.

The possible efficiency of the furnace if working at full load can be calculated from the above figures to be

$$\frac{1840-240-0.104\times1840}{1840}=\frac{1408.5}{1840}=0.7655=76.55\%$$

When a more satisfactory load factor can be secured after having overcome the troubles caused by the transformers there should be no difficulty in maintaining this efficiency, which is sufficiently high to make the working of the process satisfactory, although further progress also in this direction may be expected.

As to the power consumption of the Noble furnace, no accurate figures are at hand, but from the manager of the plant it has been stated that the power consumption has averaged 1940 kw-hours per ton. This figure has never been reached in the Trollhättan furnace.*

The cause of this lower power consumption is certainly partly due to the high-grade ore used, but the fact that gas circulation has only been used incidentally and also that the dimensions of the furnace are considerably smaller than those of the Trolltättan furnace have also contributed to the greater economy obtained.

Gas Circulation.

The idea of using circulating gas in an electric reduction furnace was originated by Harmet in 1901.

The effect of the circulating gas is a double one, namely, on one side the cooling of the charge in the neighborhood of the point of introduction of the gas and on the other side the conveying of the heat absorbed in the lower part of the furnace to the charge in the shaft.

The cooling effect becomes of paramount importance in furnaces which are worked along the line of the Trollhättan furnace and in which most of the heat is generated near the free surface of the charge in the melting chamber.

The cooling effect depends upon the amount of heat absorbed by the decomposition of H_2O and CO_2 when brought into contact with incandescent carbon, respectively, in raising the temperature of the gas.

*See, however, the very latest results obtained at Trollhättan, as recorded in our October issue, p. 482. During the week of September 3 to 9, 1911, the figure of 1736 kw-hours per ton was reached.—Editor.

To obtain an idea of the amount of heat that in this way can be absorbed, Table 2 has been computed from the average results of the Trollhättan furnace.

TABLE II.—EFFECT OF CIRCULATING GAS.

Ratio of Circulating Gas to Produced Gas.	1.		2.		3.	
Temperature to which gas is raised	500°	800°	500°	800°	500°	800°
Heat in gases in kw-hours	205°	290°	410°	580°	615°	870°
Heat absorbed in decomposition of CO_2, kw-hours	354°	354°	708°	708°	1,062°	1,062°
Heat absorbed in decomposition of H_2O, kw-hours	110°	110°	146°	146°	165°	165°
Total heat absorbed by circulating gas, kw-hours per ton of pig	669°	754°	1,264°	1,434°	1,792°	2,097°

These figures should show that a very effective cooling can be obtained for the protection of the roof, but from the great amount of heat that would be absorbed by a complete decomposition of the CO_2 it appears probable that only a comparatively small part is actually decomposed, considering that the total power consumption has only been 2391 kw-hours per ton.

For the effective protection of the cover against the heat radiation from the arcs it appears as if it should be sufficient to use a circulating gas of not more than twice the amount of produced gas.

In the case of a furnace of the Noble type, in which the electrodes enter so far into the charge as to prevent arcing, it has been found unnecessary to use any circulating gas at all as long as the electrodes remain unbroken.

It has been previously stated that the weight of the gases produced are only 35 per cent to 40 per cent of the weight of the charge, and as the specific heat is very nearly the same for the gas as for the charge the heat absorbed by the produced gas is not sufficient to heat the charge to any high temperature. In fact gases of 1000° can only heat the charge to about 350°.

This has already been recognized in 1901 by Harmet, who has in several patents proposed to increase the volume of gas by establishing a current of circulating gas taken from the top of the shaft and introduced into the melting zone by means of a blower.

The quantity of heat that can be absorbed by the circulating gas is, however, restricted by the temperature of the charge and the surface of contact of the charge through which the gas has to pass.

Without arrangements by which the circulating gas is forced to pass through the space between the electrodes and the charge, and thus through the electric arc, it is not possible for the gases to acquire any high temperature. The observations made at the Trollhättan furnace tend to show that the circulating gas in fact only serves to cool the charge and has not been able to acquire a temperature exceeding 690° C. at the neck of the melting chamber, the average value having been only 510° C.

The manner in which the circulating gas is introduced into the furnace at Trollhättan can thus not effect an essential extension of the zone within which a reduction by CO takes place.

To make the gas effective in this respect arrangements must be provided by which the gases find an opportunity to acquire a higher temperature mainly by passing the gas through the hottest part of the charge.

Load Factor.

The average power consumption of the Trollhättan furnace constructed for 1840 kw has been about 1285 kw over the whole time of operation from Nov. 15, 1910, to April 9, 1911. The average load factor has thus been only about 70 per cent.

As the load factor is of the greatest importance for the economical working of the process, it is of interest to study the variation of the same during the different periods. This is done in Table 3.

TABLE III.—LOAD FACTOR.

	Period. From	Period. To	Total Hours.	Total Kw-hours.	Average Kw.	Load Factor.
1	Nov. 16	Feb. 11	2,115.5	22,651,029	1,252	68.1%
2	Feb. 11	Feb. 19	189.5	312,601	1,650	89.7%
3	Feb. 19	Mar. 19	660.25	650,480	986	53.6%
4	Mar. 19	April 9	528.5	877,706	1,660	90.3%
	Nov. 16th	April 9th	3,502	1,500,596	1,285	69.9%

The low load factor during the first period is probably partly to be ascribed to inexperience with the manipulation of the furnace, partly to the defectiveness and the long break down of Transformer II.

The poor load factor during period 3 seems to have been due exclusively to the break down of the Transformer II. The load factors of the periods 2 and 4 approach 90 per cent, a figure which must be considered very satisfactory.

As there should be no difficulty to avoid break downs of the transformers, the experience at Trollhättan shows that an average load factor of 90 per cent can be maintained during the period of the working of the furnace, thus without taking into consideration the time for repairing and heating up.

It has already been shown that the temperature in the shaft does on an average not exceed 500° C. With suitable bricks it ought, therefore, to stand for years without any repair.

The melting chamber, which naturally is exposed to heavier wear and tear, will have to be relined from time to time. The duration of the first crucible, being about six months, must be considered as most satisfactory. The loss of time for exchange of the lining can be greatly reduced by arranging the melting chamber in such a way that it can be removed altogether and a spare crucible, ready lined, can be put in place.

With such an arrangement the total interruption from the exchange of the lining should not need to exceed a week, and can probably be reduced to two or three days. The load factor for a whole year should, therefore, not need to fall below 88.5 per cent, which means that the furnace could be worked at full capacity during 7750 hours out of a total of 8760 hours per year.

Conclusions.

The results of the Trollhättan furnace prove that the electric reduction of iron ore can be carried out in a technically satisfactory way, although the process is open to improvements by which the cost of production can be reduced.

By eliminating the water from the circulating gas the coal consumption can be reduced by about 14 per cent. The use of burnt lime instead of limestone will secure a further economy as there will be less CO_2 in the gas, which thereby will become more strongly reducing.

The average economy in power consumption which may be gained is:

By eliminating the water in the circulating gas 139 kw-hours per ton of pig, or 5.76 per cent.

By using burnt lime instead of limestone 92 kw-hours per ton of pig, or 3.18 per cent.

By heating the charge outside the furnace so as to evaporate the water 62 kw-hours per ton of pig, or 2.58 per cent.

The total average saving thus possible amounts to 293 kw hours per ton of pig, or 12.15 per cent.

Further savings in power are possible by reducing the dimensions of the furnace as much as possible and by always running the furnace at full load.

A small saving may also be secured by preventing the air from entering into the furnace. The average amount of air per ton of pig has been approximately 45 kg, burning 6 kg of charcoal and producing 25 kw hours. If the price of the current is 0.16 cent per kw hour, corresponding with $10 per h

year, the value of the heat generated is 4.4 cents, whereas with a price of the charcoal of $10 per ton the value of the charcoal burnt is 6 cents.

If provisions were made to obtain all the savings possible, the following result would have been obtained. The average coal consumption would have been 360 kg instead of 418 kg per ton of pig, and probably still lower, as a greater part of the ore would most likely have been reduced by CO on account of the stronger reducing action of the gases. The average power consumption would have been:

$$\frac{1,721.4-208}{0.765}+\frac{1,513.4}{0.765}=1,985 \text{ kw-hours,}$$

instead of 2391 kw-hours per ton.

With an average load factor of 88.5 per cent the output would have been:

3.0 tons per kw year, paid for
2.87 " " hp year " "

Even still better results may be anticipated because of the saving in power consumption through the stronger reducing capacity of the gases. As it is not possible to predict how much the reduction of the ore by CO may be increased, the saving in power-consumption from this source has not been taken into consideration.

Beckenham, Kent,
England.

Appendix I.—Analyses of Raw Materials and Total Amounts per Ton of Pig.

AVERAGE ANALYSIS OF ORE.

1.

Fe_2O_3	6.51%	Fe	62.5%
Fe_3O_4	77.92%	Si	3.82%
FeO	0.55%	Mn	0.45%
MnO	0.59%	S	0.0095%
MgO	2.66%	P	0.0115%
CaO	1.59%		
Al_2O_3	0.84%		
TiO_2	0.24%		
SiO_2	8.12%		
P_2O_5	0.0285%		
S	0.0095%		
Loss on ignition	0.942%		
	100.00%		

AVERAGE ANALYSIS OF CHARCOAL.

2.

H_2O	14.58%
Gases	10.26%
Ashes	3.06%
C	72.10%
	100.00%

AVERAGE ANALYSIS OF CHARCOAL GASES.

3.

	Per Cent. of Gases.	Per Cent of Charcoal.
CO_2	38.6%	3.96%
CO	41.65%	4.272%
CH_4	19.04%	1.955%
H	0.71%	0.073%
	100.00%	10.260%

AVERAGE ANALYSIS OF CHARCOAL ASHES.

4.

	Per Cent. of Ashes.	Per Cent. of Charcoal.
SiO_2	37.64%	1.1523%
K_2O, Na_2O	6.04%	0.1840%
CaO	19.33%	0.5920%
MgO	5.53%	0.1690%
Fe_2O_3	22.48%	0.6880%
Mn_2O_3	3.85%	0.1178%
Mn_3O_4	1.87%	0.0572%
P_2O_5	1.00%	0.0306%
SO_3	0.92%	0.0281%
CO_2	1.34%	0.0410%
	100.00%	3.06%

AVERAGE ANALYSIS OF SLAG.

5.

SiO_2	43.06%	Fe =	2.25%
Al_2O_3	5.50%	Si =	20.1%
TiO_2	2.45%	Mn =	1.48%
FeO	2.89%	S =	0.03%
MnO	1.91%	P =	0.028%
CaO	30.13%		
MgO	13.93%		
CaS	0.0674%		
P_2O_5	0.0597%		
	99.9971%		

AVERAGE ANALYSIS OF PIG IRON.

6.

Fe	95.918%
C	3.36%
Si	0.32%
Mn	0.37%
S	0.015%
P	0.017%
	100.00%

AVERAGE ANALYSIS OF DRY GASES BY WEIGHT.

7.

		C.	O.	H.	N.
CO_2	33.16%	9.06%	24.1%		
O	0.12%		0.12%		
CO	60.45%	25.9%	34.55%		
H	0.83%			0.83%	
CH_4	1.4%	1.05%		0.35%	
N	4.04%				4.04%
	100.00%	36.01%	58.77%	1.18%	4.04%

MATERIALS FOR ONE TON OF PIG.

8.

a. *Ore:*

Fe_2O_3	105.62 kg
Fe_3O_4	1,266.29 kg
FeO	8.97 kg
MnO	9.62 kg
MgO	43.25 kg
CaO	25.85 kg
Al_2O_3	13.64 kg
TiO_2	3.90 kg
SiO_2	131.95 kg
P_2O_5	0.46 kg
S	0.15 kg
Loss on ignition	15.30 kg
Total	1,625.00 kg

b. *Charcoal:*

H_2O	60.9 kg
Gases	42.9 kg
Ashes	12.8 kg
C	301.4 kg
Total	418.0 kg

Gases from Charcoal:

CO_2	16.55 kg
CO	17.38 kg
CH_4	8.16 kg
H	0.9 kg
Total	42.99 kg

c. *Limestone:*

Fe_2O_3	0.31 kg
MnO	0.45 kg
MgO	0.40 kg
CaO	70.21 kg
SiO_2	2.18 kg
P_2O_5	.....
S	0.0013 kg
CO_2	55.65 kg
Total	129.2 kg

d. *Pig Iron:*

Fe	959.18 kg
C	35.60 kg
Si	3.20 kg
Mn	3.70 kg
S	0.15 kg
P	0.17 kg
	1,000.00 kg

e. *Gases—dry:*

CO_2	281.86 kg	C	106 kg
O	1.02 kg	O	499.6 kg
CO	513.82 kg	H	10.0 kg
H	7.05 kg	N	34.4 kg
CH_4	11.91 kg		
N	34.34 kg	Total	850.0 kg
Total	850.00 kg		

Appendix II. on the coal consumption during different periods and Appendix III. on power consumption will be found on pages 638 and 639 respectively.

Electrolytic silver may be removed easily from cathode sheets by painting the sheets with a mixture of silver nitrate, copper nitrate and hydrochloric acid. After drying these sheets can be used for two or three months before deposits will begin to stick. At the San Francisco mint electrolytic refining is done with a direct current at 15 volts, and a current density of 8.3 amperes per square foot of cathode surface.

Concrete mortar blocks may be constructed according to the following method, as outlined in the *Cement Age*. The proportions of material are: 1 part by volume of portland cement, 3 parts by volume of freshly screened sharp sand, 5 parts by volume of freshly broken rock of any character suitable for concrete work, and passing a 3-in. ring. This proportion is continued to a point within 2 ft. of the upper surface of the mortar block, when the proportions may be changed as follows: 1 part by volume of portland cement, 2 parts by volume of freshly screened sand, 4 parts by volume of freshly broken rock passing a 1.5-in. ring. The upper layer of 6 in. from the top of the block should be constructed with the following proportions: 1 part by volume of portland cement and 1 part by volume of freshly screened sand. In some cases these blocks have been built for each 10-stamp battery, or in others a solid block has been built under the entire 100 stamps in the mill, without any expansion joints; and after several years' service the block appears to be in excellent condition.

Appendix II.—Investigation of the Coal Consumption During Different Periods.

Number of Charges.		1–10.	11.	12–17.	18–29.	1–29.
1. Average gas analysis by weight in per cent	CO_2	33.53	37.60	33.25	28.70	33.16
	O	0.186	0.05	.....	.....	0.12
	CO	59.45	55.10	61.60	66.80	60.45
	H	0.792	0.782	0.84	1.04	0.83
	CH_4	1.62	0.92	0.97	1.23	1.40
	N	4.475	5.56	3.34	2.01	4.04
Total	C%	35.87	34.54	36.20	37.40	36.01
Total	H%	1.397	1.012	1.082	1.347	1.18
2. Average weight of charcoal per 1 ton pig, in kg	Water	69.1	50.8	59.8	40.2	60.9
	Gases	41.7	36.9	49.3	43.1	42.9
	Ashes	11.8	11.0	13.2	17.2	12.8
	Coke	293.1	277.6	323.4	325.7	301.4
Total		415.7	376.3	445.7	426.2	418.0
3. Average weight of limestone per 1 ton pig, kg		85.2	61.4	336.0	158.0	130.2
Average weight of CO_2 in limestone per 1 ton pig, kg		36.5	26.3	144.0	67.8	56.0
Average weight of C in limestone per 1 ton pig, kg		9.95	7.18	39.3	18.5	15.26
4. Average weight of C charged in	kg					
C from limestone	kg	9.95	7.18	39.3	18.5	15.26
C from electrodes at 95.7%	kg	5.58	5.02	4.32	3.7	5.05
C from electrodes, pieces	kg	0.80	0.66	1.51	0.29	1.04
C from coke at 80%	kg	0.36	.....	.....	.....	0.48
C from charcoal—coke	kg	293.10	277.60	323.40	325.70	301.40
C from charcoal—gas	kg	17.80	15.75	21.00	18.40	18.30
Total C charged	kg	327.59	306.21	389.53	366.59	341.53
5. Weight of C in pig	kg	35.20	31.70	28.80	33.00	33.60
Weight of C in gases leaving furnace	kg	292.39	274.51	360.73	333.59	307.93
6. Total weight of gases per 1 ton of pig	kg	816	795	995	891	855
7. Weight of CO_2 from limestone	kg	36.5	26.3	144.0	67.8	56.0
Weight of CO_2 from charcoal at 38.6 %	kg	16.1	14.25	19.0	16.6	16.56
Total	kg	52.6	40.55	163.0	84.4	72.56
Weight of CO_2 in gases	kg	273.0	298.55	330.5	255.4	283.56
Weight of CO_2 from oxidation of C	kg	220.4	258.0	166.5	171.0	211.0
8. Weight of CO in gases	kg	485.0	437.5	613.0	594.0	516
Weight of CO from charcoal at 41.65%	kg	17.3	15.4	20.5	17.9	17.8
Weight of CO from oxidation	kg	467.7	422.1	592.5	576.1	498.2
9. CO/CO_2 from oxidation, by weight		2.12	1.635	3.55	3.365	3.36
CO/CO_2 by volume = K		3.33	2.57	5.58	5.28	3.71
10. Weight of O in CO_2 from oxidation	kg	160.2	187.5	121.0	124.3	153.3
Weight of O in CO from oxidation	kg	267.2	241.5	338.5	329.7	284.5
Total weight of O from oxidation	kg	427.4	429.0	459.5	454.0	437.8
11. Weight of H in gases	kg	11.38	8.05	10.75	11.97	10.1
Weight of H in charcoal—gases at 5.27%	kg	2.19	1.94	2.59	2.26	2.26
Weight of H from water	kg	9.19	6.11	8.16	9.71	7.84
12. Weight of O in water, decomposed	kg	73.52	48.88	65.28	77.68	62.72
Weight of H_2O in charge, decomposed	kg	82.71	54.99	73.44	87.39	70.56
13. Weight of N in gases, from air	kg	36.40	44.1	33.2	17.9	34.5
Weight of O from air	kg	11.25	13.6	10.24	5.52	10.65
Weight of free O in gases	kg	1.52	0.4	.....	.....	1.03
Weight of O from air, burnt by C	kg	9.73	13.2	10.24	5.52	9.62
14. Average analysis of pig in per cent =	Fe	95.69	96.27	96.78	96.93	95.92
	C	3.52	3.17	2.88	3.3	3.36
	Si	0.313	0.267	0.196	0.43	0.32
	Mn	0.453	0.270	0.101	0.316	0.37
15. Weight of O from ore at 0.3825 kg per kg	Fe	366	368	370	371	366.5
Weight of O from ore at 1.142 kg per kg	Si	3.6	3.1	2.26	4.9	3.65
Weight of O from ore at 0.291 kg per kg	Mn	1.3	0.8	0.3	0.9	1.08
Total weight of O from ore	kg	370.9	371.9	372.56	376.8	371.23
16. Weight of O from ore	kg	370.9	371.9	372.56	376.8	371.23
Weight of O from air	kg	9.73	13.2	10.24	5.52	9.62
Weight of O from H_2O	kg	73.52	48.88	65.28	77.68	62.72
Total weight of O to be reduced	kg	454.15	433.98	448.08	460.00	443.57
17. Weight of C in gases as CH_4	kg	9.95	5.58	7.25	8.20	8.97
Weight of C in charcoal gases as CH_4	kg	6.10	5.51	6.54	6.25	6.13
Weight of C for the formation of CH_4	kg	3.85	.....	0.71	1.95	2.84
18. Kg C per kg $O = 0.75 \frac{k+1}{k+2}$		0.609	0.586	0.651	0.647	0.619
19. Weight of C for different purposes:						
Weight of C to reduce ore	kg	225.5	217.5	242	243.5	229.6
Weight of C to reduce H_2O	kg	44.75	28.65	42.5	50.2	38.80
Weight of C to burn O from air	kg	5.92	7.73	6.67	3.57	5.95
Weight of C to form CH_4	kg	3.85	.....	0.71	1.95	2.84
Weight of C in pig	kg	35.20	31.70	28.80	33.0	33.60
Total weight of C required	kg	315.22	285.58	320.68	333.22	310.79
20. Weight of C actually supplied:						
Weight of C from charcoal	kg	293.10	277.60	323.40	325.70	301.40
Weight of C from electrodes	kg	5.58	5.02	4.32	3.70	5.05
Weight of C from electrodes—pieces	kg	0.80	0.66	1.51	0.29	1.04
Weight of C from coke	kg	0.36	.....	.....	.....	0.48
	kg	299.84	283.28	329.23	329.69	307.97
21. Weight of charcoal, which can be saved if no H decomposed, thus no H_2O in circulating gas	kg	69	39.0	59.6	68.2	58
Per cent of actually used		16.6	10.4	13.35	17.0	13.85

Appendix III.—Power Consumption.

Number of Charges	1–10.	11.	12–17.	18–29	1–29.
Calculation of power consumption for 1 ton of pig:					
1. Reduction of O from Fe at 4,330 calories	1,583,000	1,593,000	1,600,000	1,605,000	1,586,000
Reduction of O from Si at 6,850 calories	24,700	21,200	15,500	33,600	25,000
Reduction of O from Mn at 5,950 calories	7,700	4,700	1,800	5,300	6,400
Reduction of O from P about calories	1,000	1,000	1,000	1,000	1,000
Total calories for reduction of ore	1,616,400	1,619,900	1,618,300	1,644,900	1,618,400
2. Heat generated by O and C per 1 kg $O = 1850 \times \frac{k + 3.262}{k + 2}$ calories	2,287	2,360	2,155	3,170	2,260
3. Total calories from O in ore + C	849,000	878,000	804,000	818,000	839,000
Calories to be supplied by electricity for the reduction of the ore	767,400	741,900	814,300	826,900	779,400
4. Reduction of O from H_2O at 3,600 calories	264,700	176,000	235,000	279,000	226,000
Heat generated by O in $H_2O + C$ calories	168,000	115,300	140,800	168,500	141,700
Calories to be supplied by electricity for the reduction of H_2O	96,700	60,700	94,200	110,500	84,300
5. Calories to be supplied by electricity to expel CO_2 from limestone	36,500	26,300	144,000	67,800	56,000
6. Calories to be supplied by electricity for the evaporation of water	44,400	29,450	39,300	46,800	37,800
7. Calories for heating generated gases to 100° C. CO_2 at 22.7	6,200	6,770	7,500	5,800	6,450
O at 20.7	32	8	.....	.....	21
CO at 23.9	11,590	10,430	14,620	14,250	12,320
CH_4 at 62.5	825	457	603	686	750
H at 332	3,790	2,670	3,575	3,980	3,355
N at 23.9	870	1,052	795	418	826
Calories to be supplied by electricity for the heating of produced gases	23,307	21,387	27,093	25,134	23,722
8. Calories for heating circulating gases, assumed to be about	46,600	42,800	54,000	50,000	47,500
9. Calories to melt and superheat pig	283,000	283,000	283,000	283,000	283,000
10. Calories to melt and superheat slag at 500	102,500	112,000	390,000	229,000	163,500
11. Heat developed by O from air + C calories	22,200	30,400	22,100	11,950	21,700
Heat developed by O from air in kw-hours	25.9	35.5	25.8	13.95	25.3
12. Actual power-consumption in kw-hours	2,296	2,149	2,710	2,581	2,391
13. Power consumption in kw-hours for 1 ton of pig:					
For reduction of ore kw-hours	896.0	865.0	950.0	965.0	910.0
For reduction of H_2O kw-hours	112.8	70.9	109.9	129.0	98.5
For expelling of CO_2 kw-hours	42.5	30.7	168.0	79.1	65.3
For evaporating of H_2O kw-hours	51.8	34.4	45.9	54.7	44.2
For heating generated gases kw-hours	27.2	24.9	31.6	29.3	27.6
For heating circulated gases kw-hours	54.4	49.8	32.2	58.6	55.2
For melting pig kw-hours	330.0	330.0	330.0	330.0	330.0
For melting slag kw-hours	119.5	130.6	455.0	267.0	190.6
Total kw-hours, theoretically	1,634.2	1,536.3	2,153.6	1,912.7	1,721.4
14. Actual power consumption in kw-hours	2,296	2,149	2,710	2,581	2,391
Heat from O in air in kw-hours	25.9	35.5	25.8	13.95	25.3
Total kw-hours actually consumed	2,321.9	2,184.5	2,735.8	2,599.95	2,416.3
15. Efficiency of furnace in per cent. of electrical energy supplied	70.1%	69.9%	78.5%	73.4%	71%
16. Losses through radiation, etc kw-hours	687.7	648.2	582.2	687.25	694.9
17. Hours per 1 ton of pig	1.83	1.297	2.66	1.59	1.86
18. Losses per hour kw	376	500	219	432	373
19. Average power-consumption kw	1,252	1,650	986	1,660	1,285
20. Theoretical saving in power-consumption:					
If no H_2O is reduced kw-hours	112.8	70.9	109.9	129	98.5
If burnt lime is used instead of limestone kw-hours	42.5	30.7	168.0	79.1	65.3
If no H_2O to be evaporated kw-hours	51.8	34.4	45.9	54.7	44.2
Total saving per ton of pig kw-hours	207.1	136.0	323.8	262.8	208.0
21. Practical saving possible in kw-hours by using burnt lime and dry gas	296	195	412.5	358	294
22. Practical saving possible in per cent of total power-consumption	12.9%	9.1%	15.2%	13.86%	12.3%
23. Distribution of losses:					
Loss in transformer iron at 20 kw kw-hours	36.6	25.84	53.2	31.8	37.2
Loss by radiation at 220 kw kw-hours	402.5	285.–	585.0	350.0	409.0
Loss in water cooling at 6.6% kw-hours	151.0	141.7	178.6	170.2	157.5
Loss in copper of transformer and leads at 3.8% kw-hours	87.2	81.7	103.0	98.0	91.0
	677.3	534.24	919.8	650.0	694.7
24. Practical saving in kw-hours	161	101.2	140	175.8	138.8
If no H_2O is reduced	60.6	43.9	214	107.4	92
If burnt lime is used instead of limestone	74	49.2	58.5	74.5	62.3
If no H_2O is evaporated					
Total	295.6	194.3	412.5	357.7	293.1
25. Practical saving in per cent of total power-consumption:	7	4.7	5.16	6.8	5.26
If no H_2O is reduced	2.64	2.04	7.90	4.15	3.81
If burnt lime is used instead of limestone	3.33	2.29	2.16	2.88	2.08
If no H_2O is evaporated					
Total	12.97	9.03	15.22	13.83	12.15

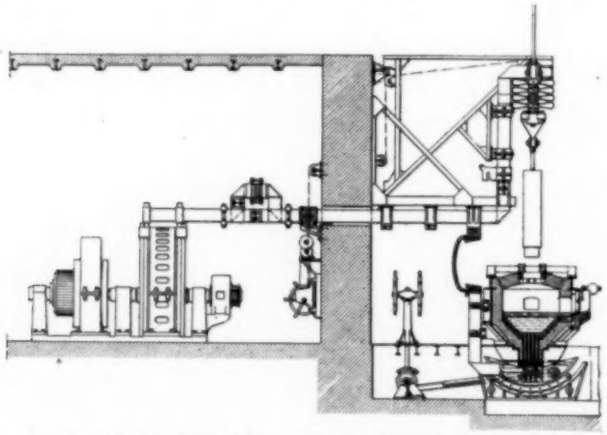
Remelting of Ferromanganese in the Electric Furnace and the Use of Molten Ferromanganese for Deoxidation.

An article by Bronn and Schemmann on the remelting of ferromanganese and the use of molten ferromanganese as a deoxidizer was abstracted in the November issue of this journal, page 575.

Another very interesting article dealing with the same subject was published in *Stahl und Eisen* of Sept. 7, 1911, page 1467, the author being Dipl. Engineer Felix Schroeder, of Livet (Isère), France.

The waste of ferromanganese when charged cold, especially during the time when slag is being removed, caused the abandonment of this process. Other disadvantages were delays due to the time required for the absorption of the solid ferromanganese, a cooling down of the bath and an irregular distribution of the manganese in the steel.

The best method for melting ferromanganese was found to be the use of the electric furnace, and the results obtained with one furnace of the Keller system* at the Burbacherhutte in Saarbrücken are described. It was adopted there after previous tests at the works of Keller-Leleux Company in Livet had shown the possibility of melting 80 per cent ferromanganese in the electric furnace without any losses of manganese.



ELECTRIC FURNACE FOR MELTING FERROMANGANESE.

The furnace seen in Fig. 1 was designed for the manufacture of steel and was run alternately on steel and ferromanganese during the last nine months. The use of molten ferromanganese was begun at once on a large scale. More than 1000 tons of ferromanganese have been remelted in the electric furnace, to be used as molten addition to all basic converter metal in the ladle for several months.

The continuous operation of the converter plant during the six weekdays required an uninterrupted running of the electric furnace in order to have hot ferromanganese ready whenever needed. On Sunday the metal usually remained in the furnace and was maintained fluid by the electric current, the consumption of electric current for this purpose being "very low." When no electric power was available the furnace was emptied and restarted on Sunday night. During the week the furnace was never emptied. Small repairs were done on Sunday, the hot metal meanwhile being poured into a ladle.

Changes of electrodes were very seldom necessary and required only a short time. The electrode was raised, the arm turned aside, and the new one, already hanging on another arm, lowered into the furnace.

Special attention was paid to the lining of the furnace, in order to avoid any interruption of the furnace operation due to repairs of the lining. The longest period during which the furnace was used for melting ferromanganese was three months. During this time the roof of acid brick, which had already been used before during a period of steel refining, was neither repaired nor renewed.

The dolomite lining was not renewed, and only a few small repairs were made from time to time. After these three months of ferromanganese melting the furnace was started to run on steel again, and the lining was still in fair condition.

The roof and lining of the furnace when used for melting ferromanganese, therefore, last at least three months, and the author claims that a lining of magnesite would stand from six to nine months in continuous operation.

The inclined lower lining between door and hearth and also the lining of the hearth itself were not changed for nine months, requiring only a few repairs.

The above facts, therefore, disprove the fear sometimes expressed that the hearth of the Keller furnace, owing to its complicated structure, would have only a short life. It has also been suggested that the iron rods of the hearth would likely melt or be converted into ferromanganese, but neither did happen, as it could be avoided very easily*

The furnace is started by lowering the upper electrode to the charge of solid ferromanganese. It is not necessary nor desirable to have previously melted iron in the furnace.

The operation of the furnace is very simple, one melter and a boy being sufficient. The best place for the furnace is the charging floor of the converter.

The molten ferromanganese is poured into a small ladle and brought to the different converters. The addition of it into the steel ladle is carried out in the same way as the addition of spiegeleisen in the production of rail steel. The furnace may be dipped by hydraulic pressure like the converters.

From the results obtained at the Burbacherhutte, the following conclusions may be drawn:

1. The remelting of ferromanganese in the Keller furnace occurs without any losses of manganese by volatilization.

For example, 10 average samples of solid ferromanganese had the following analyses before charged into the electric furnace:

80.44	81.69	82.32
80.12	79.81	79.18
79.80	79.52	79.71
80.32	Average = 80.29 per cent Mn.	

Thirty-seven successive samples of molten ferromanganese taken from the electric furnace had the following analyses:

81.35	79.82	79.18	80.32
81.06	81.06	79.18	81.06
81.38	80.12	81.06	81.38
80.44	80.44	80.75	81.06
80.12	80.12	81.06	80.44
79.81	80.44	79.50	79.81
79.20	79.50	80.12	79.50
80.12	80.32	79.91	80.32
80.11	79.32	79.51	79.91
79.32	Average = 80.22 per cent Mn.		

The remelting, therefore, causes practically no loss of manganese. There is no smoke nuisance at the furnace.

2. The net saving due to the use of molten ferromanganese amounts to 35 per cent. of the amount of solid ferromanganese which would be necessary for the production of steel of the same quality and under the same conditions of operation. Another saving results from the fact that all the fine pieces of ferromanganese, which are usually lost in ordinary practice, can be used up.

In this comparison it is assumed that the same amount of manganese shall be in the finished steel as in usual practice. As a matter of fact, however, it is possible, if the addition of

*For a detailed description of the Keller furnace see our Vol. VII, page 255 (1909).

*This would seem to contradict the experience of Bronn and Schemmann with the steel electrodes in the bottom of the Girod furnace (see page 575 of our November issue).—Editor.

ferromanganese is made in molten form, to reduce slightly the manganese content in the finished steel without affecting the rolling quality.

3. Costs of melting based upon a price of half a cent per kw-hour were approximately $5 per ton of ferromanganese, including electric power consumption, electrode consumption, wages, linings, and repairs. These are not the costs of special experiments, but a monthly average, and therefore include costs of heating and starting the furnace, delays, interruptions due to electrode changes, Sunday work, etc., or, briefly, all the expenses of using the process in a commercial basic converter plant.

Assuming that the melting-cost per ton of ferromanganese is $5, the price of one ton of ferromanganese $40 only (abroad), and that 5 kg. of ferromanganese are to be used per ton of steel, we obtain the results given in Table I.

TABLE I.

Yearly production of steel in tons.	Ferromanganese remelted in tons.	Total melting cost in dollars.	Saving of ferromanganese in tons, if 1 kg. of ferromanganese is saved per ton of steel.	Price of ferromanganese saved in dollars.	Saving of ferromanganese in tons, if 2 kg. of ferromanganese is saved per ton of steel.	Price of ferromanganese saved in dollars.
100,000	500	2,500	100	4,000	200	8,000
200,000	1,000	5,000	200	8,000	400	16,000
300,000	1,500	7,500	300	12,000	600	24,000
400,000	2,000	10,000	400	16,000	800	32,000
500,000	2,500	12,500	500	20,000	1,000	40,000
600,000	3,000	15,000	600	24,000	1,200	48,000
800,000	4,000	20,000	800	32,000	1,600	64,000

It may be said that, on account of the smaller ferromanganese consumption, the weight of steel so produced is also less. The slag is also less. The useful phosphorus in the slag increases and there is no reduction of phosphorus back into the metal. All these facts have a very small influence on the final results.

The price of ferromanganese at present is very low and may still decline. The above consumption of 5 kg. ferromanganese per ton of steel is also an assumed figure; it will in reality vary according to the quality of steel produced at different plants, therefore the above calculation should only be considered as an example.

It shows, however, that in no case in melting costs exceed the saving in ferromanganese even if only 1 kg. of ferromanganese per ton of steel should be saved. But the situation is improved considerably if 2 kg. or more of ferromanganese can be saved.

Besides this, the use of ferromanganese offers many other advantages which should not be overlooked. The desired quality of the steel is far more easily obtained than is possible with the use of solid ferromanganese. All of the molten alloy is completely absorbed by the hot metal. With some experience in estimating the manganese contained in a sample of converter metal taken after the blow, it becomes easy to determine accurately the amount of ferromanganese to be added. This is of great value for high-carbon steels.

Using solid ferromanganese on cold heats in the converter, one always runs the risk of its not being absorbed in the bath.

The distribution of molten ferromanganese in the ladle is uniform, the deoxidation quick and more effective, and rephosphorization impossible.

Not only is the quality of the steel improved, but the capacity of the converter plant can also be increased since the time required for the resting of the metal to permit the ferromanganese to dissolve is saved. The gain per heat is always a few minutes, which amounts to something when running full capacity.

There is no reason why this process should not be successfully employed in open-hearth practice. R. J. W.

The Production of Oxygen in the Workshop.

I.—The Production of Oxygen from Alkali-Chlorates.

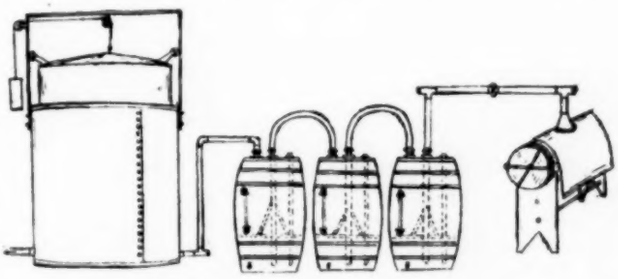
BY AUG. S. NEUMARK.

There are two ways open for the average consumer to obtain oxygen, namely, to purchase the gas compressed in tanks from the manufacturer or to make it himself by chemical means.

Among the various methods used for the production of oxygen, the one generally used (and the most abused, I might add) is the chlorate process. If potassium chlorate is heated to a temperature of 350°-370° Cent., it melts and then liberates all of its oxygen.

The decomposition is an exothermic one, *i. e.*, heat is evolved during reaction, and when it once has set in, the evolution of the gas becomes very violent. This action can be checked by adding a medium which serves to prevent too rapid generation and also lowers the temperature necessary to start the same. For want of a better name such agents are called catalytic agents; the part they play during decomposition is somewhat obscure.

Manganese dioxide (black oxide of manganese) is usually employed for that purpose and is intimately mixed with the chlorate in the proportion of 1:6 or 1:8. The addition of manganese dioxide causes the formation of small quantities of chlorine, and since pure chlorate does not liberate such, it is most likely that the manganese dioxide causes a momentary conversion into potassium manganate, thus setting free chlorine, which on further heating decomposes the manganate again, liberating oxygen and leaving as a residue a mixture of potassium chloride and manganese dioxide. The reactions may be expressed by the following two equations:



OXYGEN PLANT (COMPRESSOR TYPE).

$$(1)\ 2KClO_3 + 2MnO_2 = 2KMnO_4 + Cl_2 + O_2.$$

$$(2)\ 2KMnO_4 + Cl_2 = 2KCl + 2MnO_2 + 2O_2.$$

The admixture of the manganese dioxide reduces the temperature necessary for decomposition to 200-210° Cent.

A much better substitute would be ferric oxide, a red powder, used as a material for paints. However, it must be pure and previously have been thoroughly heated. It will reduce the temperature required to 110-120°C.

Other so-called catalytic agents, which have been recommended, are copper oxide, lead peroxide, graphite, dry quartz-sand, and powdered glass.

The addition of common table salt will further check a too violent generation. Mixtures giving the best results consist of 2 parts of chlorate, 2 parts of table salt and 3 parts of ferric oxide; or 12 parts of chlorate, 6 parts of salt and 1 part of manganese dioxide.

According to an English formula, the addition of powdered iron and a small quantity of iron oxalate will still further reduce the temperature.

Large crystals of potassium chlorate must never be used and care should be taken that the chemicals are pure and in the dry state. The presence of any chlorite or hypochlorite in the chlorate is liable to cause an explosion; barium sulphate, which has been used to adulterate the chlorate can also cause explosive decomposition.

One pound of potassium chlorate yields 4¼ cu. feet of oxygen, the cost being about 2½ cents per cubic foot.

Sodium chlorate, having a lower molecular weight, produces from 16 to 18% more gas per unit of weight, thus reducing the cost to about 2 cents per cubic foot. There is no reason to think that sodium chlorate is more dangerous to use than the potassium salt. If the conditions are alike, the decomposition of the former will be as slow and smooth as with the latter.

Violent generation is caused by the melting of the chlorate before giving of its oxygen, and then liberating all of it at once. This, however, can never happen if the materials are pure, if the chlorate is in a perfectly dry state and in form of small pieces or crystals and if intimately mixed with the catalytic substance.

The retorts used for generating oxygen must be of a proper construction. Manganese dioxide or ferric oxide is always carried along by the force of the oxygen current. It is, therefore, essential that the outlet pipe is sufficiently wide. If the outlet is too small, the gas may come off at such rapid rate as to exceed the delivery power of the exit pipe, which, to make matters worse, may be choked up from a previous charge.

Even for the smallest generating outfit, the piping should have a diameter of not less than 1 inch, preferably 1½ inches. It is advisable to have a dome or a cone-shaped exit piece, and that the exit pipe should be short, without any sharp turns, so as to offer the least resistance to the gas. "L" fittings should not be used; "T's" should be substituted, with the open end closed up by a plug. This facilitates cleaning of the pipes.

The oxygen is purified by allowing it to bubble through barrels, partly filled with water and a solution of caustic soda. The delivery pipe reaches nearly to the bottom of the barrel and is provided with some arrangement to break up the bubbles. The barrels should also have a safety pipe which not only prevents any bursting, in case of a too rapid generation, but takes care of the back-pressure when generating ceases.

The last wash-barrel is connected with the gasholder from which the oxygen is forced into storage tanks by means of suitable compressors. The latter are probably the most expensive part of an oxygen plant, although a fairly good compressor, able to produce a pressure of 250 lbs. per sq. inch, can be bought for $125.00. Those having two stages of compression are, of course, more expensive. The compressors must be water-cooled and provided with proper means of lubrication. Oil must never be used as a lubricant; a soapy solution or a diluted solution of glycerine may be used instead.

Oxygen can also be generated under sufficient pressure to force it into storage tanks which would do away with gasholder and compressor and therefore take up very little space. However, such a plant is neither economical nor are the results obtained satisfactory. The waste of gas during recharging reduces the yield of oxygen to 4 cu. ft. per pound. The purification, too, is incomplete and accidents caused by leaving the valve between retort and storage tank (or scrubber) closed are liable to happen. Several of such accidents have come to the writer's notice. Safety valves are of little use; the use of check-valves, made of non-corrosive material, might, however, minimize the possibility of such accidents.

In both the compressor type and pressure type of oxygen plant the gas is liberated by external heating of the retorts. Another type is the internally heated type, where the chlorate mixture is lighted with a match or with the aid of some ignition powder whereupon the retort is sealed. The chlorate which is always in the powdered state is mixed with some combustible substance (charcoal, hydrocarbons, cellulose, powdered metal) and an inert matter (infusorial earth, clay, sand).

Following are a few mixtures which have been used for this purpose:

100 parts chlorate of potash, 40 to 60 manganese oxide, 5 to 20 infusorial earth.

70 to 90 parts chlorate of potash, 0.9 to 8.5 coal dust (lampblack), 9.1 to 21 infusorial earth.

100 parts chlorate of potash, 20 to 40 infusorial earth, 3 to 5 powdered charcoal.

500 parts chlorate of potash, 62 manganese dioxide or ferric oxide, 4 lampblack.

"Oxygenite" belongs to this class of combinations.

The Applications of Electricity in the Metallurgical Industry of Italy.*

By R. Catani.

The development of the use of electricity in recent years has greatly benefited the metallurgical industry in many ways, as a means for driving the auxiliary machinery of metallurgical apparatus, blast-furnaces, and mixers, the hoisting and transport of raw and finished materials, and the driving of rolling-mills. The last and most important development, however, is that of furnaces dependent, for their heat, upon the thermal effects of the electric current.

The Italian metallurgical industry has also largely benefited by the introduction of electric energy, but it is proposed to confine this report mainly to an account of the furnaces and electric processes applied in Italy to metallurgical production.

The subject might perhaps have been dealt with from the point of view of the various types of electric furnaces which have been tried or are being worked at present in Italy, but seeing that this would comprise almost every kind of electric furnace, both of the arc and induction types, and that the paper would consist of a series of descriptions of furnaces such as have already lately appeared in many technical publications, it appears more desirable in the present paper to discuss the metallurgical products obtained in Italy by means of the electric furnace.

The author therefore proposes to deal first with the production of electric steel; secondly, with the smelting of pig iron by electricity; and, lastly, with alloys of iron with other metals.

I. Production of Steel in the Electric Furnace.

In Italy steel is manufactured in the electric furnace either direct from ores or from iron products, and, indeed, the first experiments in the direct production of steel from ore in the electric furnace were made at Rome by Stassano in 1898.

A. Steels Produced Direct from the Ore.—The problem of the direct reduction of steel from the ore presents difficulties of such a kind as to deter many metallurgists, however well skilled, from attempting to solve it. The numerous experiments carried out last century in the endeavor to attain this object, using coal as fuel, met with no success on a commercial scale.

On the other hand, the results of a few years' trial with the electric furnace have surpassed expectation, and it may be affirmed with certainty that, in due time, by the aid of the electric furnace, the problem of producing steel by a direct process on a commercial scale will have been overcome.

The experiments of the last century cleared up several essential points which have assisted greatly in the further development of the process. In Stassano's experiments at Rome the charge was prepared in the same manner as in previous attempts to effect the direct reduction of ore with coal as fuel. The material was first crushed, pressed into briquettes with tar and finally treated in the electric furnace.

Recent trials have shown, however, that in the electric furnace it is equally easy to treat the ore whether in a fine state or in lumps, and hence it is possible to dispense with the preparation of the raw material, which forms a not inconsiderable item in the first cost of the steel.

The early experiments of Stassano, as stated by him in 1902, led to no practical results for the reason that he adhered too closely to blast-furnace practice, both in regard to the shape of

*A paper presented at the October meeting of the Iron and Steel Institute in London.

the furnace and the process itself. In fact, the first furnace used at Rome in 1898 to 1899 was a shaft furnace in which, instead of having tuyeres for blowing, electrodes were provided for conducting electric current. As to the method of smelting, Stassano arrested the ore by means of a grating at a distance of about 20 cm above the electric arcs. The experimental furnace afterwards put up at Darfo did not differ very materially from that at Rome, nor were the results much better than those previously obtained.

Stassano finally adopted a furnace of the hearth type; the principal characteristics were that it was roofed over, and long electric arcs played between the mass of the charge and the arch of the furnace. The arcs swept the surface of the slag without passing through it; the heat, therefore, was not imparted direct by the contact with the slag or the molten mass, but passed indirectly into the bath by radiation. The electrodes had a high density, and were held in water-cooled supports. The furnace was completely closed in, and provision was made for collecting and carrying off the gases evolved during the process.

Some Stassano furnaces are fixed, others can be rotated. In some the only rotary movement is that of tilting toward the side of the tapping hole so as to facilitate tapping. Others can be rotated on their vertical axis so as to stir the bath. Single-phase current with one or more pairs of electrodes or three-phase current can be used. Stassano's experiments for the direct production of steel by means of his electric furnace may be divided into two groups, namely, those performed during the years 1898 to 1902, the results of which were published in the *Rivista di Artiglieria e Genio*, March, 1902, and those which were published in 1908 in the *Giornale del Genio Civile*.[1]

In the *first trials* only hæmatite ore of first-class quality, from the Island of Elba, was used. The chemical analyses of the material smelted, of the product obtained, and the proportions of materials used per 100 kgs. of product are given in Table I.

TABLE I.—FIRST EXPERIMENTS OF STASSANO.

Ores per cent.	Charcoal per cent.	Pitch per cent.	Limestone per cent.	Average Product.
Fe_2O_3 ..... 93.02	C 90.42	C 59.20	CaO 51.21	Fe ... 99.7
MnO 0.619	Ash 3.88	Hydro-carbons } 40.50	CO_2 ..... 43.43	C 0.10
SiO_2 ..... 3.790	Moisture. 5.70		MgO 3.11	Mn ... 0.10
S 0.058		Ash 0.27	Al_2O_3 } 0.50	Si ... 0.10
P 0.056			Fe_2O_3 .. }	
CaO + MgO .. 0.500			SiO 0.90	
Moisture 1.720				
QUANTITY USED PER 100 KG. OF STEEL PRODUCED.				
Kilog. 165.8	Kilog. 26.5	Kilog. 20	Kilog. 19.1	Kilog. 100

Altogether there were produced only 125.6 kg of steel. The loss of iron in transition from the ore to the steel was about 8 per cent. The consumption of energy averaged 4187 kw-hours per ton of product, and the mean efficiency of the furnace was 48 per cent. Stassano omitted to give particulars of the electrode consumption, but, according to an estimate of his, it was stated to amount to 12 kg per ton produced.

These experiments prove, first, the possibility of obtaining fluid steel direct from the ore in the electric furnace; secondly, the advantage that with such a furnace the amount of materials consumed approximates more closely to the theoretical amount of materials required than was the case in the older processes for the direct production of steel from ore.

The industrial importance of the process, however, was insignificant, for the reason that only material of first-class quality could be used, that the total production was negligible, that it was considered necessary to crush the ores and agglomerate them with tar and, finally, that the energy consumption was excessively high. Stassano concluded his account of 1902 with the following estimate of the cost of the direct production of steel by his processs:

[1]See also the extensive article on his work by Major Stassano in this journal, Aug., 1908 (ELECTROCHEM. & MET. IND., vol. VI, p. 315).

	Lire.
1600 kg of ore at 15 lire per 1000 kg	24.0
Crushing the ore at 3 lire per 1000 kg	4.80
200 kg of limestone at 5 lire per 1000 kg	1.0
250 kg of coke at 45 lire per 1000 kg	11.25
Crushing the coke at 2 lire per 1000 kg	0.50
190 kg of pitch at 70 lire per 1000 kg	13.30
Manufacture of briquettes at 3 lire per 1000 kg of the mixture	6.75
Consumption of electrodes (12 kg at 30 lire)	3.60
Maintenance of the furnaces	12.0
Labor	6.0
Maintenance of ingot molds and apparatus	3.0
4000 hp at 0.0057 lire per hp hour	22.80
General expenses	3.0
	112.0 = $27.62

(NOTE.—1 lira may be taken as equivalent to 19.3 cents.)

Certain of the data in this estimate appear to be inaccurate; for instance, the use of metallurgical coke is referred to, whereas charcoal was used in the experiments. Also, the cost per kilogram of electrode at 0.30 lire (6 cents) is too low, and the electrode consumption of 12 kg per ton of steel is untrustworthy.

Moreover, Stassano subtracted 18 lire from the 112 lire as representing the value of the carbon monoxide and of the volatile hydrocarbons of the pitch, the total amount of gas yielded being 900 cu. m., valued at 2 centesimi (0.4 cent) per cubic meter. But it is to be noted that, if that consideration is reasonable in the case of the electric furnace, it also holds good for the other ordinary processes in which gas is evolved from coke furnaces, blast-furnaces, and steel furnaces. Theoretically speaking, these gases would have a higher value than the gas from an electric furnace.

Hence, in making comparisons of the cost of ordinary steels, the amount of 112 lire, as given above, should be taken instead of 94, the figure obtained by deducting the value of the gas estimated at 18 lire. Further, to the figure of 112 lire ought to be added the charges for cooling water, for depreciation and for the additions in the refining process, though, on the other hand, it might be reduced by the amount of cost of preparing the ore mixture if, as in other electrical processes for the conversion of ore into steel, it should prove possible to treat pulverulent ores by the Stassano process.

In the *second series of his experiments*, Stassano used a different ore, of the following composition:

	Per Cent.
Fe_2O_3	68.70 Fe = 48.09
Mn_3O_4	3.23
SiO_2	17.15
Al_2O_3	2
CaO	1
MgO	5.67
P	0.15
S	0.12

In this case, also, the ores were crushed and briquetted, but with a 25 per cent solution of silicate of soda instead of with pitch.

In spite of the impurity of the ore, steels of good quality were obtained, the analyses of the product of charges Nos. 2 and 4 being given in Table II:

TABLE II.—STEELS PRODUCED DIRECT FROM THE ORES IN A STASSANO ELECTRIC FURNACE.

Charge No.	COMPOSITION.					MECHANICAL TESTS.		Coefficient of Quality A×R.
	C.	Mn.	Si.	P.	S.	Tensile Strength, Kg/mm². R.	Elongation per Cent. A.	
2	0.26	0.21	0.03	0.010	0.040	55	23	1,265
4	0.80	0.30	0.22	0.015	0.045	86.3	13	1,122

In the case of this second series of experiments, in which only four charges were worked, particulars were not given either of the total production or of the consumption of electrode and of water, or of the composition of the limestone and the charcoal. The furnace was of 200 hp, with three-phase current, and the average consumption of electric energy was 4250 kw-hours.

The charges were as follows:

	Kilograms.
Ore	100
Limestone	35
Charcoal	24
Aqueous solution of silicate of soda	8
Carbide of calcium	5

On the basis of these figures and of the percentage of iron in the ore, and assuming that in this second series of experiments the loss of iron was the same as before, namely, 8 per cent, the consumption of material per ton of steel produced would have been:

	Kilograms.
Ore	2,250
Charcoal	540
Limestone	787
Silicate of soda	180
Carbide of calcium	112

In addition, the author has calculated from the particulars furnished by Stassano at the Sixth Congress of Applied Chemistry, that his three-phase electric furnace of 200 hp used in these last experiments consumed, for every ton of metal produced, about 50 cu. m of water for cooling the holders of the electrodes.

B. Steels from Iron Products.—With the electric furnace, however, the general practice is to use pig iron for the production of iron and steel in the same manner as in the ordinary type of furnace, in which case the material can be charged into the furnace in either the liquid or the solid state. The consumption of electric energy diminishes rapidly the greater the amount of heat stored in the materials when charged, and the greater the degree of the purity of these. Table III shows the approximate power consumption in certain cases taken from actual practice.

TABLE III.—KW-HOURS NECESSARY FOR THE PRODUCTION OF A TON OF STEEL (RODENHAUSER-SCHOENAWA FURNACE).

	Kw-Hours.
From cold pig iron	1,500
From liquid pig iron	1,100 to 1,200
From cold pig iron and cold scrap	900 " 1,300
From liquid pig iron and cold scrap	600 " 1,000
From cold scrap	300 " 900
REFINING.	
Steel of crucible quality from liquid iron	200 to 300
Ordinary electric steel from liquid iron	120
Mixer	50

In general, where electric energy is costly, it is convenient to refine only liquid charges in the electric furnace, but steelworks are not always in the neighborhood of electric power stations of sufficient capacity to supply energy at reasonable prices. In that case it is necessary to melt solid charges in the furnace.

1. Cold Charging.—The raw materials consist generally of pig iron, scrap, ore, and additions.

Stassano Furnace.—The first installation in Italy was that of Stassano at the Royal Artillery Works at Turin, established by the Italian Ministry of War. That furnace has been in operation since 1903, and is of 200 hp. Subsequently there was formed in Italy the Stassano Electric Furnace Company, which established an electric steel works at Turin. The electrical characteristics and the number of these furnaces were as follows:

Number of Furnaces.	Horse-power.	Kind of Current.	Volts.	Amperes.
2	100	Single-phase	75	1,000
2	200	Three-phase	100	900
2	1,000	"	150	2,700

Three-phase currents at 21,500 volts were supplied by the Societa Anonima Electtricita Alta Italia, and the voltage reduced by special transformers to 75, 100, and 150 volts for the furnaces of 100, 200 and 1000 hp respectively. The single-phase furnaces of 100 hp were shunted on a single-phase circuit.

The current suplied by the above-mentioned company to the furnaces amounted to about 350 kw from December to March, and to 800 kw from March to December, while the total power of the furnaces was 2600 hp = 1910 kw, as may be calculated from the foregoing data.

The Turin works of the Stassano Electric Furnace Company manufactured chiefly parts of railway carriages and wagons, and of automobiles. The Stassano Thermo-Electric Furnace Company went into liquidation in 1909, and some of the electric furnaces of the Turin plant have been transferred to the works of the "Elba" Company at Portoferraio and some to the Milan Steel Works.

The Stassano furnace installed at Portoferraio now works with cold charges, but it is probable that as the company has available liquid pig iron and Bessemer steel it will soon adopt liquid charging for the production of high-quality steel. The installation at Milan is not yet in operation.

The results obtained with the 200-hp furnace at the Royal Artillery Works at Turin and at the Stassano Thermo-Electric Furnace Company, published by Stassano in 1908, are set forth in Tables IV to VII.

The Héroult Furnace.—Héroult furnaces are installed in Italy at the works of the Mannesmann Tube Company, formed by the amalgamation of the Mannesmann Tube Works at Düsseldorf and of the Metallurgical Company of Rome. The works are situated at Dalmine in the district of Sabbro, Province of Bergamo. Three-phase currents are delivered at the central station at 12,000 volts, where they are transformed to 550 volts. There are two furnaces in which open-hearth steel scrap is melted, and both furnaces are in operation at the present time.

The Girod Furnace.—One Girod furnace is in course of construction at the steel works at Cornigliano Ligure of the Giovanni Ansaldo Company. The furnace is three-phase, of 350 kw at 50 cycles, with a capacity of 4 to 5 tons. The material is charged cold, the maximum potential between phases being 70 volts. The furnace will shortly be put into operation, and is intended for the manufacture of special steels only. It can also be operated with liquid charges, in which case the production will be increased by about 40 to 50 per cent.

Liquid Charging.—Electric steel furnaces may be charged either with liquid pig or with steel produced in the converter or open-hearth, and either the acid or basic process may be employed. Where steel from the converter or open-hearth furnace is charged a simple refining operation alone is required, and the improved quality, and particularly the increased resistance to shock, of the electric steel appear to be due to the much lower percentage of occluded gas in that material as compared with the gas contained in ordinary steel, the commercial composition being the same in both cases.

The Kjellin Furnace.—The first furnace which was worked with liquid charges in Italy was the Kjellin furnace installed at the works of the Giovanni Andrea Gregorini Steel & Iron Foundry Company at Castro on Lake Iseo. At first cold scrap was used, but it is now working on liquid steel taken from a 4-ton open-hearth furnace, which supplies to the electric furnace 1400 kg of metal of the following composition at each heat:

Carbon per cent.	Silicon per cent.	Sulphur per cent.	Phosphorus per cent.	Manganese per cent.
0.08 to 0.10	0.03 to 0.05	0.010 to 0.020	0.008 to 0.015	0.18 to 0.35

The final product contained:

Carbon per cent.	Silicon per cent.	Manganese per cent.
0.60	0.25	0.12

The recarburization was effected by adding pig iron, and since the furnace is tapped every 2½ hours, yielding at each heat 1400 kg of steel, the daily production, reckoning in the added pig iron, amounts to about 15 tons.

The steels produced, whether of a very soft quality or hard, are very malleable notwithstanding when the manganese is

TABLE IV.—200 HORSE-POWER FURNA E AT THE ROYAL ARTILLERY WORKS, TURIN

CHARGES.			COMPOSITION OF STEEL.						MECHANICAL TESTS.		Steel Produced, Kilo-grams.	Kilowatt Hours per Ton.
No.	Composition.			C per cent.	Mn per cent.	Si per cent.	P per cent.	S per cent.	Tensile Strength, Kg/mm².	Elonga-tion per cent.		
18	Mixed scraps	350 kilograms	Maximum	0.525	1.54	0.20	0.035	0.040	83	20	11749	1250
	Scrap in bundles	200 "	Minimum	0.40	1.224	0.15	0.025	0.027	67	14	.....	.....
	Steel turnings	100 "	Average	0.451	1.388	0.176	0.030	0.033	73	16	.....	.....
	Ferro-silicon	2.50 "	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	10.50 "	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Aluminium	0.20 "	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	6 "	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ore	1 "	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

TABLE V.—200 HORSE-POWER ROTATING AND TILTING FURNACE OF THE STASSANO THERMO-ELECTRIC COMPANY (WORKING ON SOFT STEEL CASTINGS)

CHARGES.			COMPOSITION OF STEEL.						MECHANICAL TESTS.		Kilowatt Hours, per Ton.
No.	Composition.			C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.	Tensile Strength, Kg/mm².	Elonga-tion per cent.	
6	Steel scrap	330 kilograms	Maximum	0.260	0.330	0.052	0.038	1.020	45.0	19.0	.....
	Steel and iron turnings	330 "	Minimum	0.200	0.090	0.040	0.025	0.410	42.0	17.5	.....
	Steel, very rusty	70 "	Average	0.235	0.195	0.047	0.031	0.546	43.5	18.2	1260
	Haematite pig	18 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-silicon	2.2 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	6 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	30 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	4 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
8	Cast steel scrap	250 kilograms	Maximum	0.270	0.490	0.051	0.031	0.460	43.0	20.5	.....
	Sheet steel scrap	350 "	Minimum	0.140	0.100	0.040	0.025	0.390	39.8	17.0	.....
	Turnings	150 "	Average	0.207	0.236	0.043	0.027	0.421	41.2	19.0	.....
	Turnings, very rusty	70 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Haematite pig	28 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-silicon	3.43 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	5.20 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	30 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	4 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
3	Cast scrap	200 kilograms	Maximum	0.500	0.250	0.058	0.040	0.610	45.0	19.4	.....
	Sheet steel scrap	400 "	Minimum	0.280	0.080	0.045	0.035	0.360	.....	.....	.....
	Turnings	150 "	Average	0.360	0.140	0.052	0.036	0.486	.....	.....	.....
	Turnings, very rusty	70 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Haematite pig	60 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-silicon	2 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	3.60 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	35 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	5 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
9	Cast scrap	400 kilograms	Maximum	0.300	0.300	0.054	0.038	0.680	57.0	19.0	.....
	Turnings	200 "	Minimum	0.220	0.090	0.040	0.025	0.400	42.0	14.0	.....
	Turnings, very rusty	150 "	Average	0.240	0.196	0.046	0.037	0.530	48.0	17.0	.....
	Haematite pig	45 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-silicon	1 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	4.3 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	40 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	5 "	.....	.....	.....	.....	.....	.....	.....	.....	.....

TABLE VI.—1000, HORSE-POWER FURNACE OF THE STASSANO THERMO-ELECTRIC STEEL COMPANY (STEEL FOR PROJECTILES)

CHARGES.			COMPOSITION OF STEEL.					MECHANICAL TESTS.		Kilowatt Hours per Ton.	
No.	Composition.			C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.	Tensile Strength, Kg/mm².	Elonga-tion per cent.	
4	Steel turnings	2600.0 kilograms	Maximum	0.32	0.380	0.040	0.025	0.950	61	23	.....
	Soft scrap	1400.0 "	Minimum	0.300	0.150	0.035	0.022	0.780	57	19.8	.....
	Ferro-silicon	9.500 "	Average	0.310	0.242	0.037	0.023	0.877	58	21.4	958
	Ferro-manganese	21.500 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Aluminium	4.0 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Rolling-mill scale	30.0 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	60.0 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	15.0 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
4	Steel turnings	2600 kilograms	Maximum	0.470	0.085	0.041	0.031	1.300	71	18	.....
	Soft scrap	1400 "	Minimum	0.420	0.046	0.038	0.027	1.150	69	16.5	918
	Haematite pig	130 "	Average	0.450	0.061	0.039	0.029	1.211	70	17.1	.....
	Ferro-silicon	130 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Ferro-manganese	41 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Aluminium	4 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Rolling-mill scale	30 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Lime	60 "	.....	.....	.....	.....	.....	.....	.....	.....	.....
	Carbide of calcium	15 "	.....	.....	.....	.....	.....	.....	.....	.....	.....

TABLE VII.—SPECIAL STEELS. ROTATING AND TILTING FURNACES OF 200 HORSE-POWER

No.	Products.	Composition of Steel Product.					Mechanical Tests.		Kilowatt Hours per Ton.	
		C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.	Tensile Strength, Kg/mm².	Elongation per cent.		
	Nickel steel	0.180	0.123	0.048	0.007	0.780	56	29	1180	Ni = 5.60
	Nickel steel	0.165	0.110	0.015	0.007	0.690	54.5	30.1	1200	Ni = 5.56
	High speed steel	1.560	0.062	0.012	0.010	0.979	.....	.....	1150	W = 1.27; Cr = 3.5
	Tungsten steel	0.960	0.123	0.018	0.018	0.600	.....	.....	1195	W = 1.560

quite low. They are even mallable with the following extreme compositions:

	Per cent.
Carbon	0.07 to 0.68
Silicon	0.015 " 0.10
Sulphur	0.018 " 0.007
Phosphorus	0.008 " 0.006
Manganese	0.006 " 0.08

The Kjellin furnace of the Gregorini Company is a simple induction furnace taking 300 kw. It is supplied with single-phase current at 25 cycles. When working on cold charges the production is about 10 tons daily, the energy consumption in that case being 690 kw-hours per ton of steel produced. The patent furnace lining of the Poldihütte is used, which lasts for about 120 heats, during the first 80 of which exceptionally rapid working is possible. High speed steel can be manufactured in this furnace containing, say, 25 per cent of tunsgten and 6 per cent of chromium.

II. Production of Pig Iron in the Electric Furnace.

Experiments made in Italy in the smelting of pig iron in the electric furnace differ somewhat from those made in other countries with the same object. In the trials made at Sault Ste. Marie, on behalf of the Canadian Government, a full account of which was given in Dr. E. Haanel's report in 1907, iron ores were smelted in a Héroult furnace. Dr. Haanel referred in his previous report to the First Canadian Commission in 1904 to the manufacture of electric steel, in which he described his experiments with the Héroult process at La Praz and with the Keller process at Livet. That report deals only with experiments, and not with the manufacture of steel on an industrial scale, such as that which has been attempted by the Swedish Company Aktiebolaget Elektrometall, which, having ample financial resources at its disposal, has produced many tons of pig iron in the furnaces of Lindeblad, Stalhane and Grönwall. Ordinary grades of iron ore were used, both in the experiments described by Dr. Haanel and also in the smelting of pig iron on an industrial scale by the Swedish Company.

In Italy experiments have been carried out by Carcano[2] for the production of pig iron from the residues of the pyrites which abound in this country. The pyrites residues can always be used in the blast-furnace, but thorough purifying and enrichment, as well as agglomeration by special processes, are necessary to prepare them for treatment in the blast-furnace. The chemical composition of the natural pyrites residues varies considerably, but what chiefly differentiates it from ordinary iron ore is its high sulphur percentage, often exceeding 4 per cent.

The pyrites residues treated by Carcano in the electric furnace had the following composition:

	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Silicon	8.0	12.25	9.10	7.9
Iron	56.0	47.0	52.0	60.3
Aluminum	9.30	18.80	12.30	6.9
Sulphur	2.01	2.63	4.25	2.01
			Per Cent.	

The residues were charged in their natural powdery condition, and it was endeavored to work with the most basic slag possible, preferably with a multiple base and with a certain percentage of manganese, so as to ensure that the slag had a sufficiently strong desulphurizing effect. The necessity of highly basic slag has been much discussed when it is desired to obtain a very energetic desulphurizing action, but, generally, manganese is found to have a sufficiently desulphurizing influence. This was also evident in the experiments under consideration, as may be seen from the following chemical determinations made on the raw material and on the pig iron produced under these conditions:

	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Manganese	0.558	0.744	1.28	2.14	2.54
Sulphur	0.146	0.124	0.093	0.048	0.026

The production of pig iron amounted to something under 100 tons, and the quality of the product may be judged from the following analyses:

Sulphur.	Phosphorus.	Manganese.	Silicon.
Per cent.	Per cent.	Per cent.	Per cent.
0.046	0.063	2.23	3.83
0.058	0.075	2.17	4.46
0.015	0.073	2.26	6.70
0.026	0.016	2.54	5.70 carbon = 1
	Ferro-Alloys.		
0.015	0.060	11.25	3.30
.....	0.060	13.66	8.90

In sample No. 4 of the foregoing series of analyses the carbon amounted to only 1 per cent, while the manganese and silicon were much higher. The material might be classed as a special kind of steel rather than as a pig iron.

In these trials various types of electric furnace of different powers were used. The first furnace was 180 hp, while that tried in 198, which Carcano found to give the most favorable results, was of a capacity varying from 200 to 300 hp. It was a closed furnace with neutral hearth supplied with single-phase or three-phase current with automatic charging, and with recovery of the carbon monoxide evolved during the reactions. That gas was used for the purpose of preventing deoxidation of the pyrites residues.

The energy consumption per ton of pig produced averaged 2100 kw-hours in a furnace of 180 kw and using pyrites residues containing 50 to 55 per cent of iron in the form of ferrous oxide.

The lower ends of the vertical electrodes rested on the slag, so that the furnace worked as a surface resistance furnace and the slag was of very high temperature, much above that of the bath. By this means it was possible to obtain highly fluid slags sufficiently basic in character, which accounts for the ability to desulphurize sufficiently when using materials containing so high a proportion of sulphur.

As regards the economic aspect of the process, too many data are unfortunately lacking, such as electrode consumption, charges for cooling water, refractory material for lining, and labor, to make it possible to present a practical estimate. There is to be taken into account, besides, the cost of raw material and of transport, the pyrites, at least in Italy, being generally situated at a considerable distance from the regions where water-power is plentiful. On the other hand, the districts abounding in water-power are far from the markets for metallurgical products and from the seaports.

III. Ferro-Alloys.

The chief metallurgical products manufactured in the electric

[2] This journal, April, 1909 (Electrochem. & Met. Ind., vol. VII, p. 155).

furnace in large quantities are ferro-alloys. Some, such as ferro-manganese and ferro-silicon low in silicon, were already known before the introduction of electric smelting, but with the electric furnace it is possible to produce alloys with a carbon content very much lower than that obtainable when smelting these alloys in the blast-furnace. The production of alloys such as high grade ferro-silicon, ferro-chromium, ferro-tungsten is only practicable in the electric furnace, and these are manufactured either from the ores of the respective metals or direct from the metals themselves or from the metals and ores together. The only ferro-alloy electrically manufactured in Italy on an important scale is that of ferro-silicon, of which 628 tons were produced in 1909. It may be useful, nevertheless, to describe briefly some of the experiments which have been made.

A. *Ferro-Alloys from Ores.*—As already stated, Mr. Carcano has obtained a kind of spiegel and silico-spiegel using pyrites residues. The author of the present paper has made some experiments for the purpose of studying the economic possibility, under certain given conditions, of producing ferro-silicon by smelting, in the electric furnace, silicates of iron or iron ores too highly siliceous for treatment in the blast-furnace, also cast or pig iron that has been rejected on account of its high sulphur or phosphorus contents.

The electric furnace used for the experiments was one of those specially modified by the author, the special characteristics of which are the absence of any kind of flexible conductor, a minimum quantity of current passing through the conductors and, above all, through the electrodes, a minimum space taken up by the conductors and confinement of water-cooling to those parts of the furnace above the plane of the electric arcs.

As a result of careful attention to these details the furnaces, which have also been used, with suitable modifications, for the manufacture of carbide of calcium, have a high thermal and electric efficiency and consequently a high total of efficiency, whatever kind of current is employed.

With alternating current, which is now generally used in modern metallurgical works, the power factor of such furnaces approaches as near as practicable to unity. The current supplied to the furnace is one of 12,000 amp at 42 volts, 25 cycles. The production in round figures was about 20 tons of ferro-silicon of varying silicon percentages.

At first a silicate of iron was used of the following chemical composition:

	Per cent.		
Silica	25		
Ferrous oxide	56.22	Iron	43.85
Alumina	2.55		
$CaCO_3$	12.50	Lime	7
$MgCO_3$	2.10	Manganous oxide	1
Moisture and impurities	1.63		

The ratio of ferrous oxide to silica, in the specimen treated, was $56.22/25 = 2.25$.

One difficulty consisted in estimating beforehand what weight of silica would pass into the slag. Supposing that the slag formed a bisilicate, $SiO_2 + MgO$, $SiO_2 + Al_2O_3$ ($3SiO_2$), the silicon of which amounted to 28 per cent, then on the basis of that composition and the percentages of CaO, MgO and Al_2O_3 contained in the ore the quantity of silica taken up by the slag would be as follows:

$$SiO_2 = 2.55(Al_2O_3)\frac{180}{102} + 7\ (CaO)\frac{60}{56} + 1\ (MgO)\frac{60}{40} = 4.5 + 7.53 + 1.50 = 13.53\ (SiO_2).$$

So that of the 25 units of SiO_2 contained in the ore only 11.47, corresponding to 5.39 of silicon, will have remained for the ferro-silicon alloy. On the other hand, the ferrous oxide would have yielded 43 units, making a total of 48.39 units of ferro-silicon. The percentage composition of the ferro-silicon obtained would thus consist of:

	Per cent.
Silicon	11
Iron	89

According to the known formula for the reduction, by means of carbon, of ferrous oxide and silicon, and according to the analysis of the ore, the fixed carbon necessary for 100 kg of ore would amount to

$$56.22\ (FeO)\frac{12}{72} + 11.47\ (SiO_2)\frac{24}{60} = 13.94,$$

and if the metallurgical coke at disposal contained 85 per cent of fixed carbon, the weight of coke required would be

$$\frac{13.94}{0.85} = 16.4.$$

The weight of the slag per 100 parts of the ore would be:

$$2.55\ (Al_2O_3) + 7\ (CaO) + 1\ (MgO) + 13.53\ (SiO_2) + \text{ash}\ (16.4 - 13.94) = 26.54$$

In the absence of an analysis of the ash of the coke, and in consideration of its small amount, it may be assumed that the chemical composition of the slag would be that corresponding to the formula of the supposed bisilicate. Its composition would therefore be

$$Al_2O_3\ (3\ SiO_2) = 2.55 + 4.50 = 7.05$$
$$CaO,\ SiO_2 = 7 + 7.53 = 14.53$$
$$MgO,\ SiO_2 = 1 + 1.50 = 2.50,$$

and the proportion of the several constituents would be:

	Per cent.
Al_2O_3	10.6
CaO	29.1
MgO	4.2
SiO_2	56.1

According to the triangular diagram of Boudouard of silico-aluminates of calcium, the melting temperature of such a slag is about 1350° C., which is considerably lower than that of silicon, at 1900° (according to Boudouard); of alumina, at 2200° (according to Richards); and much lower than that of calcium, 3000° (according to Moissan). The slags become in fact quite fluid, and are very easily poured off and separated from the ferro-silicon.

The melting temperature will be even lower than the theoretical, since the material also takes up a certain proportion of ferrous oxide of the furnace lining and of burnt electrodes.

The charge, consisting of 100 parts of ore and 16.4 parts of coke, should yield 48.34 kg of 11 per cent ferro-silicon, and about 27 kg of slag with a silicon content of $56.1 \times \frac{28}{60} = 26.32$ per cent; whereas the average of the analyses showed that the silicon in the slag was 23.5 per cent, and in the ferro-silicon 16.5 per cent. The weight of the slag was greater than the calculated weight.

In order to increase the silicon content quartz of the following composition is added:

	Per cent.
SiO_2	91
FeO	2.6
Al_2O_3	4.10
CaO	0.9

By this means alloys of increasing percentages of silicon can be produced, up to a maximum of 25 per cent.

Still richer alloys were obtained with the same quartz, but substituting for the silicate of iron an ore of the following composition:

	Per cent.	
Ferrous oxide	75.5	Iron.....57.3
Silica	9	
Alumina	4	
Lime	0.84	
Manganous oxide	1	
Moisture and losses	9.4	

There were produced 9400 kg. of ferro-silicon. Of a 40 per cent aloy 5400 kg. were manufactured, the following materials being used per 1000 kg.

	Kilograms.
Ore	1,300
Quartz	2,330
Coke	1,000

The yield per kw-hour was 0.143 kg., and per kw-year 1.444 tons.

The electric energy was measured on the high tension connection at the switchboard, so that the pressure at the terminals

on the furnace may be taken as 5 to 8 per cent lower. The year is taken at 8000 working hours.

During these trials slags were obtained rich in silicon, even up to 38 per cent. Assuming again that such a slag was a bisilicate, the silicon by weight would have been

$$4\frac{180}{102} + 0.84\frac{60}{56} + 1\frac{60}{40} = 9.46$$

the slag by weight would have been 15.30, and its chemical composition would be $Al_2O_3 = 26.1$ per cent, $CaO = 5.4$ per cent, $MgO = 5.5$, SiO_2 61.8 per cent. The melting temperature of such a slag is at least 1500° according to Boudouard.

The remarkable quantity of quartz consumed accounts for the refractoriness of the slag. The furnace worked at a high temperature, and there was a noticeable volatilization of the silicon. Another series of results was similarly interpreted by the author. An ore was treated, consisting of the following composition:

	Per cent.
Ferrous oxide	33
Silica	56
Alumina	5
Lime	1

By means of calculations similar to the foregoing it was reckoned that the slag, without making additions, would have the composition, silica = 62 per cent, alumina = 32 per cent, lime = 6 per cent, the melting point of such a slag being above 1600°. The consumption of raw materials was somewhat higher than the theoretical amount. The ore actually contained 25.74 per cent of iron, 26.32 per cent of silicon, of which 4.65 passed into the slag. Hence for every 100 kg of ore there should be produced 25.74 + 21.67 = 47.41 kg of ferro-silicon. That is, for 100 kg of ferro-silicon, the ore consumption should amount to 211 kg, whereas instead it amounted to 356 kg.

From the results tabulated above, the efficiency of the electric furnace may be deduced. The following thermoelectrochemical data constitute the necessary factors:

Energy of dissociation.—$(SiO_2) = 180{,}000$ calories $= 3{,}000$ calories per kg. of SiO_2
$= 6{,}384$ calories $= 7.4$ kw. hrs. per kg. of silicon.
$(FeO) = 657{,}000$ calories $= 1.36$ kw. hrs per kg. of iron.

Heat Units required for Heating.—The melting temperature of anhydrous silica is about 1900°, but for the quartz used containing 91 per cent of silica the melting temperature may be assumed, on account of the impurities, to be not higher than 1800°.

Per kilogram of quartz from 0° to 1,800°, there are required 761 calories (Richards)
Per kilogram of ferrous-oxide from 0° to 1,800°, there are required 887 calories (Richards)

This latter item is based on an analogous calculation made by J. W. Richards.[1]

The mean specific heat of the carbon between 0° and 1800° is $0.5 - \frac{120}{1800} = 0.433$.

On the basis of these figures, the energy required for the production of 1 kg of 40 per cent ferro-silicon from ores may be calculated as follows:

Energy of dissociation per 0.40 of a unit of silicon	$= 0.40 \times 7.4 = 2.96$	kilowatt-hours,
and per 0.60 of a unit of iron	$= 0.60 \times 1.36 = 0.816$	"
Total		3.78 "
Heating—		
Ores	$1.3 \times 887 = 1{,}153$	calories
Quartz	$2.33 \times 761 = 1{,}773$	"
Coke	$1 \times 0.433 \times 1{,}800 = 779$	"
	3,705	"
Less—Coke, with 85 per cent. carbon, burnt to CO, yielding	$2{,}430 \times 0.85 = 2{,}065$	"
	1,640	" $= \frac{1{,}640}{864} = 1.9$ kilowatt-hours.

Total kilowatt-hours $= 1.9 \times 3.78 = 5.68$

[1] "Metallurgical Calculations," part I.

The theoretical production per kw-hour is therefore

$$\frac{1}{5.68} = 0.176.$$

The actual production per kw-hour was 0.143.
The efficiency of the furnace, therefore, equals

$$100 \times \frac{0.143}{0.176} = 80 \text{ per cent.}$$

The efficiency of the kw-year, like that of the electric, is, of course, higher, when manufacturing ferro-silicon from quartz and scrap-iron, and the use of ore or scrap is governed by the cost of the raw material and of the electric energy. In Italy, where the cost of the kw-year is generally comparatively low, it is more economical to use ore, and in order to make the question more clear the following calculations relating to the energy necessary to produce 1 kg of 40 per cent ferro-silicon, using scrap, are given for comparison with the preceding figures:

The loss of silicon is assumed to amount to 5 per cent.
Quartz required (SiO_2 90 per cent.) $= \frac{0.4 + 0.02}{0.47 \times 0.9} = 1$ kilogram.
Scrap required (Loss = 10 per cent.) = 0.66 kilogram

Energy of dissociation as above = 3.78 kw-hours. Heat units required:

Scrap	450 calories per kg. $= 450 \times 0.66 = 297$	calories
Quartz	$1 \times 761 = 761$	"
Coke	$0.7 \times 0.433 \times 1{,}800 = 545$	"
	1,603	"
Less—Carbon burnt to CO, yielding	$0.85 \times 0.7 \times 2430 = 1{,}446$	"
	157 calories = 0.18 kilowatt-hours.	

The energy required per kg. of 40 per cent ferro-silicon is therefore 3.78 + 0.18 = 3.96 kw-hours.

The theoretical production per kw-hour $= \frac{1}{3.96} = 0.252$, and the efficiency of the furnace is 80 per cent.

The actual production = 0.252 + 0.8 = 0.20 kg per kw-hour = 4.8 kg per kw-day.

Hence the consumption of energy in the ore process exceeds that in the scrap process by 1000 (5.69 — 3.96) = 1730 kw-hours per ton of 40 per cent ferro-silicon. The extra cost of this is more than saved in the cheaper raw material, especially with careful attention to the slags and to the temperatures of the thermo-chemical reactions.

With the object of reducing the slag the author has tried to produce ferro-silicon by the partial substitution of pig iron containing 94 per cent of iron for the ore. The raw materials consumed per 1000 units of the product were: ores, 520; quartz, 2200; coke, 540; anthracite, 300. With this mixture some 4000 kg of 42 per cent ferro-silicon were produced. No additions were made, and the output was low on account of the considerable loss of silicon through volatilization.

B. *Ferro-Alloys from Metallurgical Products.*—The greater part of ferro-silicon manufactured in Italy is produced from quartz and scrap iron. The first works in Italy which manufactured ferro-silicon on an industrial scale were those of the Societa Elettrochimica at Bussi. The production in 1907 was 175 tons; in 1908, 808 tons; and in 1909, 606 tons.

In 1909 the Piedmontese Company for the Manufacture of Carbide of Calcium and Allied Products produced 22 tons of ferro-silicon. According to the particulars officially published, the electric furnace for smelting ferro-silicon is absolutely identical with that used for the production of carbide of calcium, and works with the same current at the same pressure.

The author has also experimented in the production of ferro-silicon, using pig iron in place of scrap, the price of which latter material is very high in Italy. The furnace was of the same type as that mentioned in the preceding paragraph. Waste pig iron was used broken to the size of nuts. The mixture con-

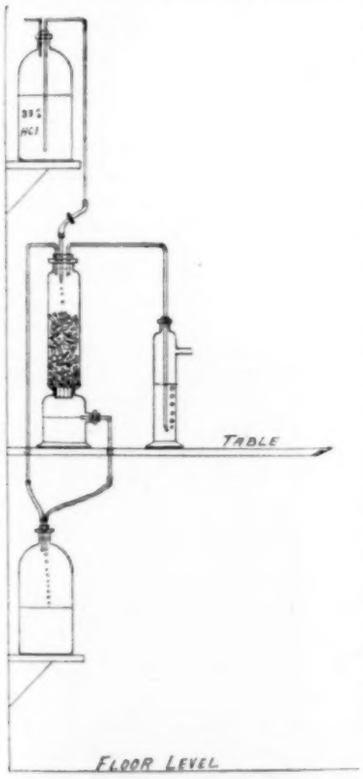
sisted of 1000 parts of pig, 3000 of quartz, and 1300 of coke. The product averaged about 65 per cent of silicon, and the total yield was somewhat under 3 tons. The raw materials used per 1000 kg of product amounted to 620 kg of pig iron, 1775 of quartz, and 810 of coke. The yield corresponded to 2.112 kg of ferro-silicon per kw-day.

Rome, Italy.

Apparatus for Making H_2S.

BY J. R. HUBER.

The accompanying sketch illustrates a convenient and satisfactory method for making H_2S in the laboratory. A Kipp generator, owing to the pressure necessary to hold up the acid when not in use, has a tendency to leak around the stoppers and consequently generates most of the time, being a continual source of annoyance besides wasting acid and iron sulphide.



APPARATUS FOR MAKING H_2S.

Hydrochloric acid, about 1 to 2, is placed in the upper bottle, the syphon filled and the flow of acid adjusted by means of a screw pinch cock. When no more gas is required the pinch cock is screwed up tight. The generation of gas ceases almost immediately.

By using an 18-in. tower the acid trickles over sufficient FeS to be entirely neutralized when it flows out at the bottom. A large cork with several small holes punched through it serves to hold up the FeS and permits the waste acid to flow into the reservoir below.

Two holes are drilled through the table top, one for the tube which carries away the waste acids and the other for a tube connecting the waste acid bottle with the top of the tower. This tube relieves the pressure in the waste acid bottle, permitting it to be tightly corked, thus preventing any odor of the gas from getting out into the room. The other bottle contains pure water for washing the gas, and after it may be placed whatever purifying or drying apparatus is considered necessary for the work in hand.

FeS in fused sticks is the most convenient form to use. The apparatus is best set up in the hood against one of the walls.

Lumen Bearing Co., Buffalo, N. Y.

Electrolytic oxygen is proposed by T. H. Aldrich, Jr., as a means of keeping cyanide solutions in an oxidizing condition. Heretofore we have depended on oxygen dissolved from the air during agitation, or on the addition of salts known to be strong oxidizing agents, for the supply of oxygen in cyanide solutions. Mr. Aldrich has found that if lead electrodes be suspended in the pulp under treatment and a current of 0.25 amp. passed through them an oxidizing condition will be maintained which effectually aids in dissolution of the gold contained in a reducing ore.

Platinum Laboratory Utensils.

At the Minneapolis convention of the American Chemical Society in a session of the Division of Industrial Chemists and Chemical Engineers, a Committee on Quality of Platinum Laboratory Utensils was appointed, which presented a preliminary report at the Indianapolis convention this summer. This committee consisted of Dr. WM. F. HILLEBRAND, of the Bureau of Standards; Dr. PERCY H. WALKER, of the Bureau of Chemistry, and Dr. EUGENE T. ALLEN, of the Geophysical Laboratory of the Carnegie Institution.

The object of the work of the committee was to study the alleged inferiority of platinum ware made in recent years as compared with that procurable many years ago.

The objections made to inferior ware are:

(1) Undue loss of weight on ignition (generally due to a content of iridium and perhaps of base metals of low boiling point).

(2) Undue loss of weight on acid treatment, especially after strong ignition (due essentially to a content of iron and calcium, which are being brought slowly to the surface and there oxidized, so as to give in extreme cases a reddish coating of ferric oxide on one hand and an alkaline reaction on the other).

(3) An unsightly appearance of the surface of the ware after strong ignition, especially in the early stages of heating (sometimes due to volatile impurities, but later caused by crystallization of the platinum, the important question arising whether this crystallization is due to the presence of an impurity).

(4) Adherence of crucibles and dishes to triangles, sometimes to such an extent as to leave an indentation of the vessel at the points of contact with the triangle, even when complete cooling has been reached before the two are separated from each other (possibly due to an inferior quality of the triangle as contrasted with the vessel).

(5) Alkalinity after ignition. (Compare above under 2.)

(6) Blistering, a defect of long standing and of somewhat infrequent occurrence.

(7) Development of cracks after continued heating, a defect "of far too common occurrence."

The committee asked for the co-operation of American manufacturers of platinum ware and received this co-operation by interviews with special representatives of these manufacturers, sent to Washington, and by the supply of platinum ware for the investigation of the committee. These samples were compared among themselves and with foreign-made ware as to behavior on heating, both as regards appearance and stiffness, loss in weight, and tests for iron and alkali. As standards for comparison were taken one crucible from Heraeus, one of his own make, of "Thermoelement" platinum, which is claimed to be free from all impurities, and three of his "Tiegel" platinum, containing, according to the latest information, not over 0.05 per cent iridium and probably less.

The committee in its preliminary reports describes the experiments made as follows:

"Surface Aspect of Ware after Heating.—Leaving out of account the discoloration due to the oxidation of iron, the surface indications shown by ware that has come under the observation of the committee at different times are not easy to describe.

"With good ware the crucible becomes uniformly affected by the blast when this envelops the whole vessel, either taking on a whitish, frosted appearance or becoming uniformly mottled inside and out with crystal plates which do not all reflect the light alike. There is great difference in the size of these plates, to almost microscopic in the case of the frost-like aspect noted from those that are easily apparent to the naked eye. Sometimes, when the plates are large, a decided blue color is imparted to the surface. The crucible of 'Thermoelement'

platinum of Heraeus which was tested showed only the frosted surface, while his 'Tiegel' platinum became in every case coarsely marked. No explanation for this difference has yet been offered, a difference that seems surprising in view of the supposed very slight differences in composition of the two.

"On poor ware the surface becomes more or less whitish, sometimes faintly brownish in places, but the discolorations are not uniformly distributed, and they often impart an unsightly appearance. As a rule, the lower part of the crucible, where it was hottest, remains bright, and from it stretch out more or less fan-shaped areas, the direction of which is affected by that of the flame, i. e., whether this was inclined or applied vertically. The areas often have the appearance of sublimates, such as are often produced from charcoal before the blowpipe. Now and then they have been seen starting sharply from a point. On direct and continued application of the flame their appearance and location may undergo change, and they may even disappear entirely.

"Surface Alkalinity.—Concerning this single phenomenon, which was first observed by Dr. H. C. P. Weber, of the Bureau of Standards, before the committee was organized, little definite information has thus far been gained. When manifest it can be detected by applying moist red litmus paper to the surface of the vessel after it has been blasted for several hours. It has been noted in varying degrees of intensity and is not always apparent. The element producing the alkalinity is calcium. Its sources might be lime from the crucibles used for melting the platinum or a lime salt used in polishing. Opinions as to this were conflicting, and the committee has not yet enough evidence at hand to warrant an expression of opinion of its own. The question is to be tested in part by comparison of ware that has been polished with Vienna lime and that which has not been so treated. What effect, if any, the presence of lime has on the quality of the ware is not known, although the observations of Mylius and Hüttner[1] and of Hillebrand,[2] have direct bearing on this point.

"Iron Content.—Rouge is sometimes used for polishing ware, and steel enters into its manufacture at more than one stage. These may account for a portion of the iron content of all ordinary ware, and for the minute traces that may occur in the specially refined ware, but the evidence all goes to show that the introduction from this source (unless possibly when hot rolling of sheet and hot shaping of utensils are practised) accounts for but a small fraction of the amount actually present.

"The committee has found that the 'Thermoelement' and 'Tiegel' platinum of Heraeus are satisfactory with respect to iron. It is either not detectable after long blasting or gives such a faint reaction as to represent a negligible quantity. One American firm has submitted what is claimed to be its own refined ware, which seems to be with respect to iron the equal of the 'Tiegel' platinum of Heraeus.

So far as the experience of the members of the committee goes, all the ware supplied by the American manufacturers as of their own make during recent years, has been strongly contaminated by iron. Sometimes its amount has been so great as to reveal itself after less than a half hour's exposure to a Bunsen burner flame, by a darkening, even reddening, of the inside of the crucible. More often the color change becomes pronounced only after long blasting. In an extreme case as much as 6 mg. of iron oxide were removed from the surface of a 22-gram dish of German make some years ago by one of the committee members. An amount less than 1 mg. in weight suffices to impart a very dark color to a crucible of 20 cu. cm. capacity. Even when the iron does not reveal itself thus, treatment with pure hydrochloric acid or with alkali pyrosulphate will often do so. Renewed heating will invariably bring more iron to the surface of the platinum if originally present. If the heating is done electrically instead of over a blast flame the color change due to iron oxide appears outside as well as inside the crucible. In one case a few years ago a dark color due to iron oxide was observed on ware from the maker.

"Loss in Weight on Ignition.—Aside from a content of volatile base metals, iridium is supposed to be the volatile constituent of platinum which contributes chiefly to the observed loss of weight on strong ignition. This belief is supported by the experience of the committee.

"The following set of tests on the ordinary commercial ware submitted by one firm is perhaps typical in general for a grade of ware that is supposed (from the statements made to the committee) to carry about 0.05 per cent iridium, but further confirmative tests must be made before final conclusions are reached.

"A crucible, weighing without cover about 16 grams, lost in four hours over a vertical blast 3.4 mg. The temperature inside the crucible and near the bottom was 1180°-1190°C. The rate of loss per hour was 0.85 mg. At 1100° the loss in less than four hours was 1.2 mg., or at the rate of somewhat over 0.3 mg. per hour.

"Another crucible of the same lot, but slightly heavier, suffered a loss over the blast in 3 hours at 1200°-1210° of 2.3 mg. (0.77 mg. per hour) and of 1.9 mg. in addition at 1100° in 4.5 hours (0.42 mg. per hour). The surface of the crucible was very unevenly and strongly frosted, in marked contrast to the crucible of refined metal.

"Other tests by one of the committee, on purchased domestic ware, showed losses of the same order, in some cases fully 1 mg. per hour at or near 1200°.

"Contrasted with these, the 'Tiegel' platinum of Heraeus above referred to shows to decided advantage. A crucible weighing about 16 grams lost in 12 hours (heatings of 4 hours each) 2.3 mg. (0.2 mg. per hour) at temperatures not exactly known but approximating 1200°. At 1100° the loss was 0.9 mg. in 8 hours (0.11 mg. per hour). Another lost at the same rate as the first in 8 hours at 1190°-1210°.

"Marked improvement was shown also by crucibles made of metal specially refined in this country, free from iron, and submitted by one firm, the rate of loss per hour being 0.3 mg. at 1200° in 12 hours, and 0.1 mg. at 1100° in 8 hours. The losses in the several periods of heating at 1200° were irregular, due perhaps to marked variations in atmospheric conditions, causing errors in weighing.

"The crucible of 'Thermoelement' platinum lost in 14 hours at 1200° 2.3 mg. (0.17 mg. per hour), and in 4 hours, at 1100°, 0.2 mg. (0.05 mg. per hour).

"The triangles furnished with the crucibles, and of the same composition, suffered corresponding losses.

"The last test shows that at the temperatures employed and over a blast-lamp even platinum that is claimed to be free from iridium has a perceptible vapor pressure or forms a volatile oxide, and it is also apparent that for chemical use the 'Tiegel' platinum seems to be quite as satisfactory in quality.

"The foregoing data may be summarized as follows:

	Rate loss per hour in mgs.	
	1200°	1100°
Ordinary ware (0.5 (?) per cent Ir.)..	0.85	0.3
'Tiegel' platinum.....	0.2	0.11
'Thermoelement' platinum.....	0.17	0.05
American refined ware.....	0.3	0.1

"These data are not to be taken as rigidly exact, but they are probably fairly indicative. It should be remembered that the crucibles were not polished or wiped between the separate heatings.

"From the above and other tests it may be inferred also that there is a sharp rise in the volatility curve for iridium, and apparently for platinum also between the temperatures 1100° and 1200°. This point it is the intention of the committee to investigate further. It is true, it would seem that for temperatures up to 1100° the volatility loss is of little significance with refined

[1] Ztschr. Instrumentenkunde, 30, 190 (1910).
[2] Bull. 422, U. S. Geol. Survey, p. 122.

ware, but that it may be of some moment with ordinary chemical ware low in iridium and free from volatile base metals. At higher temperatures and with long-continued heating, however, the losses are appreciable even with the purest ware and are very high with commercial ware.

"In this connection it is proper to say that in all probability the maximum temperature of the blast used in the above experiments is above that commonly attained in most laboratories. The gas used was artificial. Attention may be called also to the fact that relatively few substances obtained in chemical analysis require blast ignition and that blasting of an hour's duration is seldom necessary, so that in the majority of cases the volatility loss may be ignored.

"Relative Stiffness and Hardness of Ware.—Pure platinum is soft and lacks stiffness. For these reasons it has been common practice to leave a certain amount of iridium in ware made from it or even to add to the amount already present. This addition is especially pronounced in triangles, which may contain 15 to 20 per cent of iridium. The crucibles of 'Tiegel' platinum made up in this country and also those of American-refined metal showed lack of stiffness in a very objectionable degree. In contrast with them was the crucible of 'Thermoelement' platinum which, though not so stiff as the crucibles of commercial ware, was sufficiently so for all practical purposes. The reason for this difference is not yet apparent, but it may be due to the different shape of the crucibles or to a different thickness of the metal. If so the defect mentioned in the American crucibles tested is one that can be easily remedied by the makers.

"Triangles.—In order that no error might be introduced by the use of triangles differing in composition from the crucibles, whereby iridium or other metal might pass from the former to the latter, the makers were asked to furnish with the crucibles triangles of the same composition, and this they did.

"The refined crucibles and triangles were found to stick together badly while hot, owing to the welding property of pure platinum. They could be separated when cool, but often only by the expenditure of some force and with danger of tearing out metal from the crucibles. It is probable that, in order to help maintain as far as possible the purity of a vessel of highly refined platinum, it should not be used in conjunction with a platinum triangle of different quality. Sagging, to a much greater degree than with ordinary triangles that have been some time in use, was not observed. The sticking defect is serious enough to take into consideration with refined ware. Therefore, experiments were made to test the temperatures obtainable inside a crucible with triangles of different materials and shapes. Without entering into details, it may be said that with a vertical blast flame the differences were far less than had been expected between triangles of platinum, of clay and of quartz, so that except for particular uses platinum triangles seem to have little advantage over those of clay or quartz when used under the conditions of the tests. The clay triangles with projecting points on which the crucibles rest allow as high a temperature to be reached in the upper part of the crucible as those of platinum. Quartz triangles when made in one piece are to be had, but the cheaper ones of nickel or 'nichrome' covered with quartz tubing are probably just as serviceable.

"A word of caution may be added here with regard to the use of 'nichrome' in direct contact with platinum. It has not been shown that there is no contamination from this source, and in view of the cost of platinum ware any additional source of contamination is to be guarded against.

"Blistering.—The blistering of crucibles seems to be far less frequent now than it was some years ago. It is probable that this defect resulted from past methods of manufacture rather than from other causes, and the committee has not given experimental attention to the subject.

"Cracking—One manufacturer attributed cracking not to the faulty nature of the ware but to careless handling of magnesium-ammonium phosphate, etc., and to heating with reducing flames. Another treated the ware mechanically to lessen the tendency to crack.

"Extra Cost of Refined Ware.—The refining of platinum to the extent shown by the 'Tiegel' platinum of Heraeus and by the recent product of one of the American manufacturers necessitates a higher charge than we have been accustomed to pay for commercial ware. In the experience of those who have bought the refined ware in this country the extra cost is in the neighborhood of 30 cents per gram. This is a large relative increase, but not excessive in comparison with the advantage gained from having an article that one knows to be free from all objectionable impurities in significant amounts. If platinum can be refined to the extent of removing all or nearly all iron and other base metals without removing all the iridium, the presence of the latter might be no disadvantage when extreme temperatures are not called for. The matter is important, for stiffness is essential to large vessels, and this quality iridium undoubtedly imparts to platinum ware. That iron can be so removed without removing the iridium the following experience of one of the committee at the Bureau of Standards appears to show. For the sake of completeness data not relevant to the point in question are not included.

"Two crucibles, No. 65, bought in 1905, and No. 117, bought from another firm in 1908, but never used before, were first heated electrically at 1250° for several hours. No. 65 showed then no iron, or at most a trace, and lost weight subsequently over the blast at 1100°-1200° at the rate of 0.64 mg. per hour. Crucible No. 117 yielded 1.4 mg. iron oxide after the electrical heating. After cleaning, subsequent blast ignition caused a loss at the rate of 1 mg. per hour for the first 3 hours and of 0.9 mg. for the next 8 hours; the surface became strongly alkaline and showed more iron oxide. From the stiffness of the crucibles and their high loss in weight it is practically certain that both contained iridium, yet the one was free from iron, the other heavily charged. The original source of crucible No. 65 is not known."

The conclusions and recommendations of the committee are as follows:

"Although it were premature to offer anything final in the way of recommendations, the investigation has brought out certain facts and observations which the committee has deemed it wise to lay before the chemists in the foregoing pages without waiting for completion of the inquiry. It is hoped that further tests on domestic and foreign ware and a closer study of the literature will enable it to present a more finished report later.

"It has been shown that ware of foreign make can be had against which nothing can be brought on the score of iron or other base metal content and in which the loss caused by strong blasting seems to be solely of platinum itself and to be entirely negligible, for all ordinary purposes, up to at least 1100°.

"It appears also that American-refined platinum ware can be had of nearly, if not quite, as good quality, and there does not seem to be any reason to think that its quality cannot be still further improved if there is a call for it to be done.

"In order to obtain such ware it is only necessary to *specify that it must show no marked uneven discoloration on heating and must give no test for iron after prolonged ignition (two hours may be regarded as sufficient) and that the rate of loss per hour at 1100° C, over a period of not less than four hours, and preferably longer, shall not exceed, say, 0.2 mg.* The makers are granted a fair margin of safety in this latter item. Ware which does not meet these conditions should not be accepted if the purchaser is willing to pay the enhanced cost of the better ware.

"These specifications are subject to revision."

The Institute of Industrial Research, of Washington, D. C., of which Dr. Allerton S. Cushman is the director, announces the organization and equipment of a Division of Roads and Pavements under the supervision of Mr. Prevost Hubbard, formerly chief chemist in the Office of Public Roads in the Department of Agriculture.

Flow of Heat Through Bodies.

By Carl Hering.

In electric furnaces the economy of the heat losses is of far greater importance than in fuel furnaces, on account of the greater initial cost of the energy. In fact the cost of the energy is in many cases the criterion, hence a mere reduction of the losses may at times make the difference between commercial success or failure. It, therefore, becomes important in such cases to be able to proportion and design the walls of the furnaces so as to reduce these losses to the least that the particular conditions warrant.

To do this intelligently requires a knowledge of the physical laws and constants pertaining to the flow of heat through bodies and from surfaces. The term flow of heat is here used because it is a general one, which refers to all the heat, whether it travels by conduction, convection or by radiation, or any combinations of these; hence it is not necessary to specify the particular way, thereby avoiding ambiguities and errors, such as the very common one of calling the heat conducted through the walls "radiation" losses, instead of conduction losses, even when this heat is taken up by a water jacket, in which case there is no radiation whatsoever.

Although the chief laws concerning the flow of heat are fairly well understood, at least for conduction and radiation, our knowledge of the physical constants is still deplorably meager. The heat conductivities of many materials have been determined, but generally not for the ranges of temperatures occurring in furnaces, and often not for the very materials for which they are needed most in electric furnaces. Moreover, many of them are of questionable accuracy and they are rarely if ever stated in the most convenient way for furnace calculations, nor in the most convenient units. And as for the physical constants for the emission of energy from surfaces to the air, water, etc., including therefore radiation, convection and true conduction (if any), existing data is still more meager, vague and unsatisfactory.

The purpose of the present article is to give a compilation of such physical constants as the writer has been able to gather from various sources, and to present them in the form in which they are most convenient to use by the electric-furnace engineer. The writer does not vouch for their correctness and in fact doubts the accuracy of many of them; he assumes responsibility only for the reduction to the units here used. They are given here merely as a collection of existing data for the benefit of those who wish to use them as they are, and with the hope that such a table will be of sufficient interest to induce others to contribute other data, corrections or more reliable figures.

From the nature of furnace walls it is far simpler to use resistances and resistivities (specific resistances) in the calculations, instead of the older and more common conductances and conductivities (specific conductances). Hence the physical constants have here all been reduced to resistivities, in which form they are not generally given in books.

The units here used are those proposed and recommended by the writer some time ago, as by far the simplest to use in calculations, namely the watt, the centigrade degree and the thermal ohm. The watt (energy per second) is used for the flow of heat in preference to the calorie per second, because all power and energy calculations of the electrical engineer are preferably made in watts and watt-hours (or kilowats and kw-hours), hence this avoids troublesome reductions; moreover, it is an international unit. The centigrade degree is used because it is the more rational scale, is international, and because most thermal data in tables is given in that scale; moreover, it is, in the opinion of the writer, the moral duty of every progressive engineer to do what he can to discourage the use of that senseless, awkward, irrational and unfortunate inheritance, the Fahrenheit scale, which the more progressive nations have abandoned many years ago, if indeed they ever used it.

Lastly, the thermal ohm is used here (for the unit of thermal resistance), because it greatly reduces the calculations, in fact makes them the simplest possible. This new unit was proposed and recommended by the writer about a year ago.* Briefly described, the thermal ohm is that thermal resistance through which 1 watt of heat flow will pass when the temperature drop is one centigrade degree. Hence, if R is the thermal resistance in thermal ohms, W the flow of heat in watts and T the temperature in centigrade degrees,

$$W = T/R$$

which might be termed the thermal Ohm's law.

Moreover, if r is the specific thermal resistance, (resistivity) in thermal ohms per centimeter cube or inch cube, then

$$R = r\,L/S$$

in which L is the length and S the cross section of a thermal conductor, either centimeters or inches being used consistently throughout. This again is like the similar formula for electric resistance. Thermal resistivities, however, seem to have high temperature coefficients and the temperature ranges are great; the resistivities of refractory materials are said to decrease at the higher temperatures; this should be taken into consideration by using the modified value of r for that range of temperature when that data exists, or to allow for it when it is not known.

It will be seen, therefore, that by using these units no conversion factors are necessary, as all coefficients become unity. For the calculations of the electrical engineer the tabulated values of physical constants are therefore best reduced once for all to thermal ohms.

To reduce a thermal conductivity in gram calories per second to the resistivity in thermal ohms, when both are for 1 cm. cube, or for 1 in. cube, multiply the reciprocal of the conductivity by 0.238882. To reduce units in terms of 1 cm. cube to those for 1 in. cube, multiply by 0.393700 when they are resistivities, and by 2.54001 when they are conductivities. To reduce thermal conductivities given in terms of British thermal units (b.t.u. or pound Fahrenheit units) per hour, per square foot surface, per inch thickness, per degree Fahrenheit, to thermal ohm, inch cube units, multiply the reciprocal of that number by 273.013. To reduce gram calories per second to watts, multiply by 4.18617. To reduce a drop of temperature in Fahrenheit degrees to the drop in centigrade degrees, multiply by 5/9. For further reduction factors see the writer's paper above cited, and his Conversion Tables.

The original data for some of the figures in the accompanying table gave the conductivities relatively to silver, water or air, hence required the absolute values of these three materials to reduce them to thermal ohms. For this purpose the conductivity of silver in gram calories per second, centimeter cube units, at ordinary temperatures was taken at 1.0; that for water at room temperatures as 0.0013; and that for air at 0°C. as 0.0000553 (Winkelmann); those for silver and air being the reduction figures recommended and used in the Landolt & Boernstein tables, and that for water being approximately the one there recommended.

Figures concerning such mobile bodies as water and air are reduced here from those given but without any endorsement of the rationality of such data, because with such mobile fluids it would seem that some transfer of heat by convection (bodily movement of the heated material) must have taken place when the figures were determined; and in gases there must have been some radiation also. Even if these other ways of the transmission of heat had been eliminated during the test, they would be present in the practical apparatus to which these figures are applied. Moreover to the writer's knowledge there may be the equivalent of a relatively very high resistance at the surface of contact of a solid and a gas or liquid, hence unless this was eliminated when the determinations were made the results obtained may be far wrong. Hence, in the writer's opinion, fig-

*"Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho," this journal, Vol. 9, January, 1911, p. 13.

ures for such mobile materials are at best only very crude, if indeed they have much value at all.

A number of values have been included here merely for the sake of comparisons, and because published values of thermal conductivities are sometimes stated relatively to some other material, and if the absolute value of any one of them is known they can all readily be reduced to their absolute values. The writer has constructed furnaces which are virtually cold on the outside, hence low-temperature insulating materials then become available; but in such cases the designer should be cautioned against the dangers of over-insulation, when the material cannot stand a possible increase of temperature.

Another table giving figures for the flow of heat from surfaces will be published in a subsequent article.

The sources of the original data from which these values were deduced are indicated by the letters in the last column, which refer to the notes following the table.

Although the data in this table are arranged to be in the best from for calculations of electric heat, they can also readily be used for steam heat, refrigeration, etc., by first completing the simple calculation in terms of the watts flowing through the insulation and then converting those watts into horse-powers, b. t. u. per hour, calories per hour or second, etc., as desired, by means of a single multiplication. Watts $\times$ 0.00134111 = horse-powers. Watts $\times$ 0.0568776 = b. t. u. per minute. Watts $\times$ 0.014332) = kg-calories per minute.

TABLE OF THERMAL RESISTIVITIES.

APPROXIMATELY IN ORDER OF RESISTIVITY.

Temperatures in Centigrade Degrees.

	Thermal Ohms. Inch Cube.	Centimeter Cube.	Reference.
Silver, 0°-100°	0.094	0.24	LB
Copper, (electrode mean), 100°-197°	0.090	0.23	H
Copper, (electrode mean), 100°-837°	0.11	0.27	H
Copper, 0°-100°, about	0.11	0.27	LB
Copper	0.13	0.32	LB
Copper, cast	0.12	0.29	CI
Copper, rolled	0.11	0.28	CI
Copper, rolled	0.13	0.32	WF
Aluminium, 0°-100°	0.27	0.69	LB
Graphite, Acheson, (electrode mean), 100°-390°	0.28	0.71	H
Graphite, Acheson, (electrode mean), 100°-914°	0.32	0.82	H
Brass, 0°-100°	0.36	0.92	LB
Iron, (electrode mean), 100°-398°	0.43	1.1	H
Iron, (electrode mean), 100°-1245°	0.49	1.3	H
Iron, wrought	0.22	0.55	CI
Iron, wrought, 0°	0.46	1.2	LB
Iron, wrought, 275°	0.76	1.9	LB
Iron, wrought	0.79	2.0	WF
Iron, cast	0.26	0.66	CI
Iron, cast, 30°	0.63	1.6	LB
Steel	0.24	0.60	CI
Steel	0.81	2.1	WF
Steel, various	0.81	2.1	LB
Steel, 10% manganese	3.0	7.7	LB
Platinum	0.25	0.63	CI
Platinum, 18°-100°	0.55	1.4	LB
Platinum	1.1	2.9	WF
Carbon, (electrode mean), 100°-942°	0.72	1.9	H
Carbon, (electrode mean), 100°-360°	1.05	2.7	H
Lead	0.33	0.83	CI
Lead	1.10	2.8	WP
Lead, 0°-100°	1.2	3.0	LB
Plumbago brick, about 1000°	3.8	9.6	WQ
Carborundum brick, about 1000°	4.1	10.3	WQ
Mercury, 0°-50°	5.5	14.1	LB
Quartz, 0°	5.9	15.	LB
Graphite, (probably plumbago), 7°	8.0	21.	LB
Retort carbon, 0°	9.1	23.	LB
Magnesia brick, about 1000°	13.	34.	WQ
Stone, calcareous, fine	16.	42.	P
Chromite brick, about 1000°	16.	42.	WQ
Ice	16.	42.	LB
Boiler scale, 60°	17.	44.	LB
Marble, fine grained, gray	9.8	25.	P
Marble, coarse grained, white	12.	31.	P
Marble, 30°	19.	48.	LB
Stone, calcareous, ordinary	20.	51.	P
Fire brick (probably room temp.)	21.	53.	D
Fire brick, about 1000°	22.	57.	WQ
Fire brick, mean for 500°-1300°	23.	57.	Z
Fire brick, mean for 0°-1300°	30.	77.	Z
Fire brick, about 400°-800°	44.	112.	CE
Fire brick, mean for 0°-500°	67.	171.	Z
Checker brick, about 1000°	24.	61.	WQ
Gas retort brick, about 1000°	25.	63.	WQ
Slate, 94°	26.	67.	LB
Building brick, about 1000°	29.	72.	WQ
Glass pot, about 1000°	35.	89.	WQ
Porcelain, 95°	38.	96.	LB
Feuerstein	39.	99.	LB
Terra cotta, about 1000°	41.	104.	WQ
Chalk (solid?)	43.	109.	LB
Cement, Portland, neat, 35°	44.	110.	N
Cement, Portland, 90°	132.	336.	LB
Lava	47.	120.	LB
Silica brick, about 1000°	47.	120.	WQ
Kieselguhr brick, about 1000°	52.	133.	WQ
Baked clay, brickwork	57.	140.	P
Building brick wall, average, 8" to 40" walls	62.	160.	W
Water, room temperature	72.	180.	LB
Glass, 28°	87.	220.	LB
Plumbago, 20°-155°, 26.1% solid matter	96.	240.	O
Fine sand, 20°-155°, 51.4 solid matter	109.	276.	O
Coarse sand, 20°-155°, 52.9% solid matter	110.	280.	O
Cork (solid?)	131.	333.	LB
Plaster of Paris, 0°	105.	266.	LB
Plaster of Paris, 20°-155°, 36.8% solid matter	221.	562.	O
Slag concrete, 34.3 lbs. per cu. ft., 50°, 1 pt. by weight slag and 0.61 cement	178.	453.	N
Pumice stone, 18.2 lbs. per cu. ft., 50°	169.	430.	N
Pumice stone	187.	477.	LB
Pumice stone, 20°-155°,34.2% solid matter	219.	558.	O
Brick dust, sifted	204.	518.	P
Asbestos, 20°-155°, 8.1 solid matter	139.	353.	O
Asbestos, 36 lbs. per cu. ft., 600°	166.	422.	N
Asbestos, 36 lbs. per cu. ft., 50°	221.	562.	N
Asbestos, with air cells	416.	1060.	S
Card board, below 0°	239.	606.	LB
Ebonite, 48°	251.	637.	LB
Petroleum, 13°	265.	672.	LB
Wood pine, parallel to fiber	313.	796.	LB
Many liquid compounds (hydrocarbons, oils, etc.), about	313.	796.	LB
Anthracite coal	317.	803.	LB
Chalk, 20°-155°, 25.3% solid matter	332.	844.	O
Very porous slag, 22.5 lbs. per cu. ft., 50°	356.	905.	N
Zinc, white, 20°-155°, 8.8% solid matter	398.	1010.	O
Fire felt, 21°-175°	418.	1060.	B
Gas works breeze	440.	1120.	C
Infusorial earth (fossil meal), 21°-175°	415.	1050.	B
Infusorial earth 20°-155°, 11.2% solid matter	435.	1110	O
Infusorial earth, 20°-155°, 6% solid matter	472.	1200.	O
Infusorial earth, burnt, 12.5 lbs. per cu. ft., 450°	263.	675.	N
Infusorial earth, burnt, 12.5 lbs. per cu. ft., 50°	477.	1220.	N
Infusorial earth, loose, 21.8 lbs. per cu. ft., 350°	427.	1090.	N
Infusorial earth, loose, 21.8 lbs. per cu. ft., 50°	562.	1430.	N
Infusorial earth	745.	1890.	C
Magnesia, carbonate, 85%, 20°-188°, average	537.	1370.	S
Magnesia, calcined, 20°-155°, 28.5% solid matter	160.	407.	O
Magnesia, calcined, 20°-155°, 4.9% solid matter	544.	1380.	O
Magnesia, calcined, 20°-155°, 2.3% solid matter	554.	1410.	O
Magnesia, calcined, 21°-175°	572.	1450.	B
Charcoal, pine, 20°-155°, 11.9% solid matter	494.	1260.	O
Charcoal, from leaves, 11.9 lbs. per cu. ft., 100°	537.	1370.	N
Charcoal, from leaves, 11.9 lbs. per cu. ft., 50°	603.	1530.	N
Charcoal	723.	1840.	C
Feathers, 20°-155°, 2% solid matter	577.	1470.	O
Sawdust, 13.4 lbs. per cu. ft., 50°	614.	1560.	N
Sawdust	620.	1570.	C
Sawdust, compressed	765.	1950.	LB
Cork, granulated and compressed, 20°-188°	467.	1190.	S
Cork, ground, 10 lbs. per cu. ft., 200°	614.	1560.	N
Cork, ground, 10 lbs. per cu. ft., 50°	797.	2030.	N
Air, 20°-155°	143.	364.	O
Air, 0°	1700.	4320.	LB
Cotton wool, 20°-155°, 1% solid matter	596.	1520.	O
Cotton wool, 20°-155°, 2% solid matter	659.	1670.	O
Cotton wool, 5.05 lbs. per cu. ft., 100°	572.	1460.	N
Cotton wool, 5.05 lbs. per cu. ft, 50°	627.	1600.	N
Cotton wool	830.	2110.	C
Cotton wool, loose	2170.	5500.	LB
Cotton wool, compressed	2810.	7120.	LB
Hair felt, 20°-155°, 9.2% solid matter	633.	1610.	O
Hair felt, 21°-175°	790.	2010.	B
Hair felt	865.	2200.	C
Hair felt, below 0°	1080.	2740.	LB
Lampblack, 20°-155°, 5.6% solid matter	697.	1770.	O
Fine quartz sand	718.	1820.	LB
Silk, 6.3 lbs. per cu. ft., 100°	662.	1690.	N
Silk, 6.3 lbs. per cu. ft., 50°	752.	1920.	N
Wool, sheep's, 20°-155°, 2.1% solid matter	616.	1570.	O
Wool, sheep's, 8.5 lbs. per cu. ft., 50°	676.	1720.	N
Wool, sheep's, 8.5 lbs. per cu. ft., 100°	745.	1890.	C
Wool, sheep's	803.	2050.	N
Mineral wool, 21°-175°	737.	1870.	B
Mineral wool, 0°-18°	1010.	2570.	C
Hard rubber	1060.	2680.	LB
Wood, pine, radially	1070.	2720.	LB
Loose fibrous materials, 9°	1540.	3920.	LB
Flannel	2650.	6720.	LB

B—George M. Brill. *Trans.* Am. Soc. Mech. Eng., XVI, p. 827. Coverings on 8-in. steam pipes.

C—J. J. Coleman. *Engineering*, Sept. 5, 1884, p. 237. Ice melted in cube surrounded with the materials. Temperatures 0-18° and 0-38° C. The values were given relatively to each other; to reduce them to absolute measure it is here assumed that the value for sawdust is 620, thermal ohm, inch cube units.

CE—Clement & Ely.

CJ—Calvert & Johnson. Relative values based on silver. Reduced here on the basis that the conductivity of silver is 1.0 in gram calories per second, centigrade, centimeter cube, units.

D—Depretz. Hood "Warming and Ventilating Buildings," p. 249. Given relatively to marble, here assumed to be 10 thermal ohms, inch cube, units.

H—Carl Hering. "The Proportions of Electrodes for Furnaces." (Table.) Paper read before the Am. Inst. Elec. Eng., March 31, 1910. Mean values when materials are used as furnace electrodes.

LB—Landolt & Boernstein tables. The values here chosen are mostly approximate means of the generally numerous and sometimes greatly differing values given by different observers. For the individual values and for the authorities, see those tables. They also includes values for very many other materials.

N—Wilhelm Nusselt. *Zeit. Ver. Deut. Eng.*, June, 1908, p. 906, table p. 1006. Materials were placed between two concentric metallic spheres or cubes. Heat generated electrically in interior. Temperature measured with thermo-couples at numerous depths in the material after several days' heating. As here given they represent the resistivities at the temperatures stated, not the means over a range. Probably the best and most reliable determinations published. His conductivities are here assumed to be in terms of kilogram-calories per hour, centigrade, meter cube, units; although not so stated directly in the original, it is undoubtedly

what is meant. An abstract appeared in the *Eng. Digest*, August, 1908, p. 168, in which the units are reduced to thermal units, feet, inches and Fahrenheit degrees; the formula there given omits to say that it is necessary to multiply by the temperature also.

O—Prof. Ordway. *Trans.* Am. Soc. Mech. Eng. Vol. VI, 1884-5, p. 168. Tested in plates 1 in. thick between two flat iron surfaces, one of them heated by steam, the heat emitted by the other being measured calorimetrically. Extended, carefully made researches; presumably very good values. There is an error in the heading in table VII, square inch should read square meter, as in the others.

P—Peclet. Box "Practical Treatise on Heat." Presumably ordinary weather temperatures.

S—H. G. Stott. Power 1902. Pipe coverings. 200 ft. of 2-in. pipe heated electrically to constant temperature. Coverings were somewhat over 1 in. thick; they are here reduced to 1 in. Heat transmitted to air, hence these resistances include that at the surface.

W—Wolff. *Jour. Frank. Inst.*, 1893. The transmission of heat from the interior to the exterior of buildings through the walls; hence ordinary weather temperatures. Prescribed by law by German Government for heating plants. Said to agree well with good American practice. The value here given is an average of all the individual ones, omitting the first one, which differed greatly from all the others.

WF—Wiedemann & Franz; relative values based on silver. Reduced here on the basis that the conductivity of silver is 1.0 in gram calories per second, centigrade, centimeter cube, units.

WQ—Wologdine, Queneau. This journal, September, 1909, p. 383. The temperatures were about 1000° C.; the materials were those of commerce and do not refer to extra pure nor to inferior grades. The present writer is of the opinion, based on the method used in the tests, that these values are probably too low.

Z—Source lost, but probably fairly good values. See this journal, February, 1909, p. 72.

Temperature Coefficient.—Very little data seems to exist on the changes of the thermal resistivity with the temperature. It is generally recognized, however, that thermal insulators decrease in resistivity quite appreciably with increase of temperature. Nusselt, in the researches above cited, paid especial attention to this and finds from his measurements that as a general rule the thermal conductivity of such thermal insulating materials increases at the rate of 1/273 part for every degree centigrade above 0°C. This is of interest also because if this rate continues below 0°C. it means that the curve goes to the absolute zero, which would mean that such materials become absolute thermal insulators at absolute zero (when heat no longer exists.) On the other hand electrical conductors are believed to conduct without resistance at absolute zero.

Iron and Steel Institute.

In our last issue an account was given of the October meeting of the Iron and Steel Institute in London with abstracts of part of the papers. We herewith give abstracts of those papers which had to be omitted in our last issue on account of limitation of space.

Oxy-acetylene Welding.

A paper by Mr. F. Carnevali is entitled "Autogenous Cutting and Welding of Metals." He deals solely with the oxy-acetylene process. The advantages of this process are pointed out and the most important conditions for securing good results are summed up as follows:

1. The use of the purest acetylene obtainable.

2. The use of a blow-pipe so designed as to insure careful adjustment in the proportions of the mixed gases, and to secure their exit at a velocity capable of maintaining the metal sufficiently fluid without the melting flame being too rigid.

3. The provision of an absolutely neutral zone in the melting flame.

4. The use of a pure conducting material of well-defined composition, similar to that of the metal to be melted, in the form of small rods, of such a size as to be rapidly melted without hindering the simultaneous heating of the adjacent parts.

5. Care should be taken that the edges should be free from impurities, and that, whenever possible, they should be arranged at an angle to each other, so as to allow the neutral zone of the flame to penetrate well into the inside, and so completely melt the two surfaces before the extraneous metal is introduced.

6. The use of deoxidizing and fluxing powders for the purpose of eliminating the oxides present, together with the incidental impurities. In view of the unavoidable oxidation of the metal subjected to the melting process, it is necessary to bear in mind the relation between the temperature of the melting points of the oxides and that of the metal itself.

7. Rapidity in melting, in order to avoid excessive heating, which will not only alter and deteriorate the original structure of the metal, but would even favor the occlusion of gas—particularly hydrogen—and so occasion the formation of blow-holes in the melted zone.

In addition to these conditions, care should be taken to see that no sudden cooling should take place owing to any sudden projection of the flame, while, on the other hand, the conditions must, of course, be modified with regard to the conductivity and specific expansion of material as well as with regard to the thickness, size and shape of the pieces operated upon.

In the author's investigations the precautions sketched above were observed and his investigation extended chiefly to the changes in chemical composition and mechanical properties caused by the welding process and to methods of avoiding such changes.

The first two parts of the paper deal with his experiments on the oxy-acetylene method of welding of mild steel and of steels containing medium and high percentages of carbon. While the mild steels and the higher carbon steels are discussed under separate chapters, the results for both classes of steels are summed up together as follows:

1. The violent heating and the sudden melting undergone by the metal alter profoundly its quality. This alteration becomes specially manifest by an internal strain of the metal, meaning by this word an injurious modification of the metal's mechanical properties, caused by the thermal and mechanical phenomena, owing to the high temperature of the process, as the stress and hardness tests show, although uninfluenced by modifications in the chemical composition of the steel. The absence of deformation in the single structural constituents in stress tests is a further confirmation.

2. The alteration in the mechanical qualities (tenacity and elasticity) cannot be remedied by simple hammering on the molten zone—which often proves to be most injurious to high-carbon steels—but must be coupled by proper thermal treatment, such as annealing, which is so much more efficient when prolonged over a sufficient time (30 or 40 minutes), so that the interior stress can be nearly wholly eliminated. Further, the other thermal processes (quenching, etc.) have the greatest influence on high-carbon steels.

3. A remarkable influence on the mechanical properties (more especially of the later steels) is to be attributed to cooling processes, the slowest cooling being the best and most effective.

4. Apart from the alteration in mechanical properties (the principal of which is the change in the elongation), very important modifications take place in the chemical constitution, especially in high-carbon steels. These changes consist chiefly in the diminution in the percentage of carbon, silicon, and manganese. Such variations, which are proportional to the percentages originally present in the steel, are, it is very easy to understand, caused by the strong oxidizing influence to which the metal is subject during the melting, and become modified by the different methods employed in this process. It is easy to remedy this defect by using as a cementing material a steel of analogous composition to the pieces intended for welding, but containing higher percentages of the more easily oxidized elements.

5. The mechanical properties are, owing to the short duration of the process, only slightly altered when, instead of the fusion process, a simple heating with the flame is substituted, as is easy to foresee. As the metal during the melting is subjected to a strong internal stress, with every increase in the depth of the fused portion, the efficiency of the remedial heat treatment, undergone after the melting, rapidly decreases.

The author then investigated the oxy-acetylene welding method of **pig iron**, the experiments being limited to grey varieties.

The results for pig iron are summed up as follows:

1. The rapid melting and cooling of the pig iron subjected to the oxy-acetylene process cause, as might have been foreseen, a fine compact structure in the zone of fusion.

2. Owing to the high temperature and to the energetic conditions of heating which take place during the process of welding, a composition of the pig iron (and therefore, of its internal structure as well) undergoes considerable modification, chiefly owing to the loss in carbon and silicon (both of which strongly influence the structure of the metal). This loss, of course, varies with the different melting processes employed. An easy method for partly remedying this loss is to employ an appropriate cementing material which will supply the elements eliminated by oxidation.

3. The mechanical properties of the metal are not greatly influenced by oxy-acetylene welding, when properly carried out. The internal stress suffered by the metal confers great hardness on the zone of fusion, as shown by the Brinell tests. The brittleness tests, although they reveal the good quality and homogeneity of the metal, do not appear to be affected by the slight variations due to the melting of the metal.

4. The heat treatment and the cooling conditions, though they exert considerable influence on pig iron subjected to the process, do not appreciably modify the mechanical properties, and are of interest only when it becomes necessary to take into consideration the dilatation of the metal in relation to its low thermo-conductivity in the welding of special engine parts in which the physical properties are of great importance.

Phosphates in Basic Slag.

A paper by Mr. VICTOR ADOLPH KROLL, of Luxemburg, gives a detailed account of the author's extended "Researches on the nature of the phosphates contained in basic slag derived from the Thomas-Gilchrist (basic Bessemer) dephosphorization process." It is a very long paper accompanied by analytical tables and some fine microphotographs.

The annual production of basic slag is about 2,600,000 tons. Th slag contains nearly one-fifth of its weight of phosphoric acid, combined in a peculiar form the exact nature of which has hitherto defied the closest investigations.

After a very complete and careful history of former investigations on this subject, the author gives an account of his own researches. He distinguishes the following phosphates in basic slag.

First, "carnotite," discovered by Carnot, the typical silico-phosphate, in form of flat prisms, probably belonging to the monoclinic system, or in form of prismatic crystals as slender needles; the formula of carnotite is

$$(3\ CaO,\ P_2O_5;\ 2\ CaO,\ SiO_2)_3$$

Second, "hilgenstockite," the tetraphosphate of Hilgenstock, in the form of yellowish flattened prisms, with the formula

$$(3\ CaO,\ P_2O_5;\ CaO)_3$$

Third, "steadite," the silico-phosphate of Stead and Ridsdale with the formula

$$3\ [(CaO)_3,\ P_2O_5];\ 2\ (CaO),\ 1\ (CaO)_2,\ SiO_2$$

This salt is therefore neither tribasic nor tetrabasic, but intermediate between the basicity of the two, and shows by its constitution that in all these slag phosphates it is not the basicity which counts, but the sum total of the foreign salts and bases linked up with the orthophosphate molecule.

"The silico-phosphate, which only contains tribasic acid, is perfectly equivalent to the tetraphosphate, because it is not the addition of a base, and a base alone, which confers a special constitution on a phosphate, but it is to a much greater extent the addition of a neutral silicate which produces the new and sufficiently stable molecule while preserving absolutely the same nature as the basic salt in which the silicate is found badly enough replaced by lime."

The theoretical part played by the tetraphosphate vanishes before that played by the neutral silico-phosphate, the apparent part played by the higher basicity being nothing more than a mere error of observation.

This composition of steadite assures to it considerably more influence in the liquid slag than its predecessors possessed. As a matter of fact, the slag in practice seldom reaches the percentage of silica and of lime corresponding to the ratio $5CaO,\ P_2O_5;\ SiO_2$, which is peculiar to carnotite, nor yet to so small a percentage of silica as to yield simultaneously hilgenstockite in the solution. Artificial enrichment accompanied by synthetical chemical reactions is therefore necessary in order to form these two constituents, and it is the selective crystallographic action occasioned by slow cooling which causes this phenomenon.

On the other hand, the composition of steadite more closely approaches that of the ratios found in the slag, and it is formed preferentially when the percentage of the latter in silica is not sufficient to give rise preferentially to more acid compounds, nor the cooling sufficiently slow to afford the crystallization a choice of selection as to the form it will take on assuming the solid state.

The main object of the author's researches, however, was to find a fourth phosphate in basic slag, intermediate between steadite and carnotite, the presence of which was sought for on the grounds of theoretical deductions. It is called "thomasite" and was found in all the slags collected from different districts.

"Thomasite" is nearly always found in the form of homogeneous cryptocrystalline masses, which may occasion difficulties in its identification. The reason is that it forms the mother-mass of the slag itself, and all the substances previously described are nearly always the products of selective crystalline decomposition, which occurs during slow cooling in the heart of an originally homogeneous mass of thomasite.

As slow cooling is just the condition necessary for obtaining fine crystals, a well-crystallized mother-mass is less frequently encountered than the products of its decomposition, which are always sought toward the center of the blocks of slag, whereas the main phosphoric mass is altogether cryptocrystalline, and is passed over unnoticed by those who are seeking to find crystals.

It is indeed its character of a mother-mass which has led to its escaping the notice of those who are in search of crystals. On a closer search, however, it can be found in all basic slags no matter what their source, and it will then be seen that it forms a very considerable cluster, out of which appear, in favorable localities, little crystals of greenish-blue color.

Thomasite is a silico-phosphate and a double salt of lime and of ferrous oxide. The formula given by the author for thomasite is

$$6\ CaO,\ P_2O_5;\ 2\ FeO,\ SiO_2$$

The phosphate in this is hexabasic, and therefore intermediate between the tetraphosphate and the octobasic salt. The reasons for speaking of hexaphosphates are discussed at considerable length.

Italy.

In view of the original intention of the Institute to hold the meeting at Turin, Italy, and to combine with it visits to the principal centers of the iron and steel industry of Italy, a large number of the papers prepared for the meeting deal with the present status of the Italian iron and steel industry.

A paper by Mr. CATANI, giving an interesting sketch of the present developments in the electrometallurgy of iron and steel in Italy, is published in full elsewhere in this issue.

A paper by D. DOMPE, of Milan, and Mr. F. S. PUCCI, of Milan, deals with "the present state of the metallurgical industry of Italy." The first part, by Mr. Dompe, gives statistical data on the production of pig iron, wrought iron, and steel, and brief descriptions of the principal works and their equipment. The second part, by Mr. Pucci, discusses the further treatment of the iron and steel and its manufacture into various products, a list of the different plants being given in a geographical arrangement according to provinces of Italy.

A paper by Mr. A. Ciampi, of Florence, deals with the iron ore deposits of Central Italy; a page by Prof. G. La Valle, of Rome, with the iron ore deposits of southern Italy and Sicily; a paper by Mr. L. Testa, of Igkesias, with the iron ore deposits of Sardinia, and a paper by Cavalier G. Calvi with the production of iron ores in the Brembana Valley.

Finally a paper by Dr. L. Leigh Fermor, of Calcutta, is to be mentioned which discusses the origin of the iron ores of Swedish Lapland.

A Continuous Carbon Dioxide and Temperature Recorder and Its Application to Combustion Efficiency *

By Ed. A. Uehling.

After the possibility to utilize the pressure of steam as a motive force had become fully recognized, the first problem was to construct a practical machine through which the heat energy stored in steam under pressure could be converted into mechanical motion. In the solution of this problem the plain slide-valve engine was evolved. With the advance of the mechanic arts this type of engine was improved in power and effectiveness.

Meanwhile improvements in the construction of the steam boiler had made higher steam pressures possible. To utilize the greater expansive force thus made available the automatic cut-off was invented, compounding was resorted to, and by adding the condenser the expansive force of steam was utilized to the practical limit. By virtue of these strides in the progress of steam engineering the consumption of steam per horse-power hour was reduced from an average of 35-40 lb. down to 12-15 lb. For a while it seemed much more could not be expected.

Then came the steam turbine. Its rapid development in the direction of high-power units caused a revolution in the application of steam as the motive force in generating electricity, and is making heavy inroads on the domain of the reciprocating steam engine in the propulsion of ships.

Compactness of construction, directness of action, avoidance of all reciprocating strains, absence of large rubbing surfaces, elimination of lubricating oil in contact with the steam, absence of cylinder condensation, are all points of superiority of the steam turbine as a prime mover.

In the direction of economy the turbine is not superior to the highest type of reciprocating engine; it excels, however, in utilizing the lower ranges of the expansive force in steam, and by combining the high-pressure reciprocating engine with the low-pressure turbine a steam consumption of 10 to 12 lb. per horse-power hour is being practically achieved.

Hand in hand with the progress in the engine room went of necessity improvements in the boiler room. In the direction of fuel economy we early find the feed-water heater and later the economizer. Every effort was directed toward increased capacity and higher steam pressure; this necessitated increased strength and resulted in the development of the water-tube boiler.

In the direction of mechanical equipment wonderful strides have been made in the past two decades. Mechanical stokers, forced-draft automatic coal and ash-handling machinery, etc. are now found in the boiler houses of all the larger up-to-date power plants. All these vast improvements in the development of the application of steam power have been accomplished by the mechanical engineer with the aid of the physicist and metallurgist, but without the direct assistance of the chemist.

This is deplorable and accounts for the fact that in utilizing the steam, which is a physical and mechanical problem, an economy of 60 to 70 per cent has been achieved, whereas the generation of the steam, which is a fundamental chemical problem, nothing worth mentioning has been gained in the past generation, 30 to 40 per cent of the heat energy stored in the coal still passing up the chimney, due mostly to lack of control of the process of combustion.

The possibilities in this direction have been shown by the brilliant investigations of Professor Bone; but while we may readily concede that surface combustion may ultimately prevail, we must not for that reason relax in our efforts to improve existing methods, and this can be accomplished only by the aid of the chemist.

I venture to say that in this country to-day, not one in a hundred of the managers, engineers or superintendents of steam-power plants knows the composition of the coal he is burning; not one in ten knows its contents of fixed carbon, volatile matter, ash, and moisture. Not in one plant out of a hundred are the products of combustion systematically sampled and analyzed. Yet in the composition of the flue gas lies the key to fuel economy.

Of the heat generated from the coal burned in the average steam plant less than 70 per cent, and in many cases not even 60 per cent is transmitted to the water in the boiler; the bulk of the balance is carried off by the product's combustion. To reduce this loss to a minimum we must know its cause, and this can be determined only from having a complete knowledge of the chemical composition as well as the physical character of the fuel used, and a continuous control over the process of combustion.

I lay special stress on the continuity of control over the process of combustion, because the conditions are continully changing, not only from day to day, and from hour to hour, but from minute to minute. Changes in atmospheric condition as to pressure, temperature, and humidity; changes in the physical structure of the coal as to preponderance of lumps or fines; changes in the condition of the fire, due to accumulation of ashes, formation of clinker, etc., and, last but not least, changes in the rate of combustion necessary because of variable steam consumption.

The best results are obtained by securing complete combustion with a minimum excess of air, and since there is true relation between the per cent of CO_2 in the flue gas to the excess of air supplied we have in this constituent an index to efficient firing. If, therefore, the per cent of CO_2 be continuously revealed to the fireman it will serve as a guide and enable him to control his fire so as to obtain the most economical results.

If we know the composition of the fuel the loss of sensible heat up the chimney can be readily calculated from the stack temperature and the per cent of CO_2 contained in the flue gas. By basing the calculation on a unit weight of the total carbon contained in the fuel the formula for computing the waste heat from combustion become very simple.

A number of instruments have been devised which automatically sample and analyze the flue gas and record the per cent of CO_2 it contains at short intervals. There is also one which does this continuously and at the same time continuously indicates the per cent of CO_2 at or near the boiler front where the fireman can see it without leaving his post, and I have been honored with the privilege to bring this instrument to your attention to-night. (See the article on p. 681 in this issue.)

Formulæ for Determining the Heat Carried Off by the Flue Gases.

NOTATION.

A = weight of air supplied in burning 1 lb. of carbon.
Ac = " " " required to burn 1 lb. of carbon.
Ah = " " " " " " the available hydrogen per pound of carbon.
Ae = weight of air supplied in excess of that required to burn a weight of fuel containing 1 lb. of carbon.
An = weight of dry gas contained in the flue gas per lb. of carbon burned.
Aw = weight of water vapor in atmosphere per lb. of air supplied.

*A paper presented at the joint meeting of the New York Sections of the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry, on November 10.

Ov = weight of volatile oxygen contained in fuel per lb. of carbon.

Ht = weight of total hydrogen contained in dry fuel per lb. of carbon.

$Ha = Ht - \frac{Ov}{8}$ = weight of available hydrogen in fuel per lb. of carbon.

$Wh = 9/8\ Ov$ = water of hydration in fuel per lb. of carbon.

Wm = weight of water (moisture) in fuel per lb. of carbon.

P = per cent of CO_2 contained in flue gas.

Pc = " " CO " " " "

L = B.t.u. loss due to sensible heat in flue gas per lb. of carbon when burned with dry air.

Lc = B.t.u. loss due to fuel value of CO contained in flue gas per lb. of carbon burned.

Lv = B.t.u. loss due to moisture in air supplied per lb. of carbon burned.

Lh = B.t.u. loss due to water of hydration in fuel per lb. of carbon burned.

Lm = B.t.u. loss due to moisture in fuel per lb. of carbon burned.

T = temperature of flue gas on leaving the boiler.

t = temperature of air in boiler house.

$S = 0.23$ = specific heat of flue gas.

$Sw = 0.48$ = specific heat of water vapor.

Pure air contains	by weight	by volume
Oxygen	23	21
Nitrogen	77	79

One cu. ft. of air at 62 deg. F. weighs .0761 lb.

" " " " nitrogen at 62 deg. F. weighs .0738 lb.

Complete Combustion of Carbon with Theoretical Amount of Air.

One lb. of carbon requires for complete combustion $Ac = 11.6$ lb. of dry air, which measures 152 cu. ft. at 62 deg. F. The products of complete combustion of 1 lb. of carbon, with the theoretically required amount of air, will weigh $Ac + 1 = 12.6$ lb. Hence the loss of heat up the chimney will be:

$$L = 12.6\ (T - t)\ S \qquad (1a)$$

Substituting the value of S

$$L = 2.9\ (T - t) \qquad (1)$$

Complete Combustion of Carbon with an Excess of Air.

Air is composed of 21 per cent of O_2 and 79 per cent of N; hence, according to Avogadro's law, 21 per cent of CO_2 is the maximum obtainable.

When carbon is burned with an excess of air (Ae), which by volume equals $\frac{Ae}{0.0761}$ cu. ft., then

$$P = \frac{152 \times 21}{152 + \frac{Ae}{0.0761}}$$

from which we get $Ae = \frac{21 \times 11.6}{P} - 11.6$

The total weight of air $A = Ac + Ae$. Substituting the values of Ac and Ae we have

$$A = \frac{21 \times 11.6}{P}$$

$$L = (1 + A) \times (T - t)\ S \qquad (2a)$$

Substituting the values for A and S and reducing to the lowest terms we have

$$L = \left(0.23 + \frac{56}{P}\right) \times (T - t)$$

Calculation of Heat Lost in the Flue Gas When Hydrogen Forms an Appreciable Part of the Fuel.

In the chemical reaction which constitutes the combustion of hydrogen, 8 lb. of oxygen and 1 lb. of hydrogen unite to form 9 lb. of water vapor. This requires 34.8 lb. of air and results in 35.8 lb. of combustion product. In commercial flue-gas analyses no account is taken of the moisture, hence only the nitrogen derived from the combustion of hydrogen effects the analytical result. The burning of 1 lb. of hydrogen with the theoretically required volume of air therefore results in 26.8 lb. of N_2 by analyses.

Hence if fuel consisting of carbon and hydrogen is burned with the theoretically required air,

we have $P = \frac{11.6 \times 21}{11.6 + 26.8\ Ha}$

if excess air is used

we have $P = \frac{11.6 \times 21}{11.6 + 26.8\ Ha + Ae}$

from which we get

$$Ae = \frac{11.6 \times 21}{P} - 11.6 - 26.8\ Ha$$

$$An = Ac + 0.77\ Ah + Ae$$

Substituting and reducing

we have $An = \frac{11.6 \times 21}{P}$

$$L = (An + 1) \times (T - t)\ S + 9\ Ha\ (T - t)\ Sw \qquad (3a)$$

Substituting the values of An, S and Sw and reducing

we have $L = \left(\frac{56}{P} + 0.23 + 4.3\ Ha\right) \times T - t \qquad (3)$

Loss Due CO in Products of Combustion.

When the per cent of CO (Pc) is known, the heat loss due to its presence (Lc) is easily determined. The weight of carbon in a molecule of CO_2 is the same as in a molecule of CO, and it follows from Avogadro's law that the weight of carbon burned to CO is to that burned to CO_2 as the per cent of CO is to the per cent of CO_2 contained in the combustion products.

Thus, letter X = the weight of carbon burned to CO,

we have $\frac{X}{1 - X} = \frac{Pc}{P}$ from which we get $X = \frac{Pc}{P + Pc}$ = carbon burned to CO per unit of carbon contained in the fuel. Since CO burned to CO_2 per unit of carbon develops 10,150 B. t. u.

we have $Lc = 10{,}150 \times \frac{Pc}{P + Pc} \qquad (4)$

Loss Due to Water Vapor in the Air.

Atmospheric moisture exerts a chilling effect on the fire because of its high specific heat and because at least part of it will be decomposed. But since under normal conditions the hydrogen which was set free by decomposition will reunite with the oxygen from the air in the combustion chamber and will evolve as much heat there as was absorbed in the fuel bed in setting it free, there results no heat loss due to this double reaction. It will be necessary, therefore, to consider only the sensible heat carried off by moisture contained in the air, hence

$$Lv = Av \times (An + 0.23\ Ah)\ Sw \times (T - t) \qquad (5a)$$

Substituting and reducing we have

$$Lv = Av\ \left(\frac{117}{P} + 0.88\ Ha\right) \times (T - t) \qquad (5)$$

All natural solid fuels contain water in two forms; moisture (Wm), i. e. water which can be driven off by drying at about 212 deg.; and water of hydration (Wh), which passes off at much higher temperature with the volatile combustible matter, the estimation of which is ordinarily but wrongly included in the volatile combustable matter.

Loss Due to Water Hydration.

If we know the weight of oxygen contained in a given fuel per lb. of carbon get the water of hydration.

$$Wh = 9/8\ Ov$$

Hence $Lh = 1.125\ Ov \times (T - t)\ Sw$ (6a)

Substituting the value of Sw

we have $Lh = 0.54\ Ov \times (T - t)$ (6)

Loss Due to Moisture.

Moisture in fuel (Wm) exists as water, which must first be raised to 212 deg., then evaporated, the vapor passing off at the temperature of the flue gas.

Hence $Lm = Wm\ (212 - t) + 966 + (T - 212)\ Sw$ (7a)

Substituting the value of Sw and reducing

we get $Lm = Wm\ [(0.48\ T) + (1076 - t)]$ (7)

Recapitulation.

(1) Heat in B.t.u. carried off by the products of combustion per lb. of carbon burned when there is no excess of air.

$$L = 2.9\ (T - t) \quad (1)$$

(2) Heat in B.t.u. carried off by the products of combustion per lb. of carbon burned with an excess of dry air.

$$L = \left(\frac{56}{P} + 0.23\right) \times (T - t) \quad (2)$$

(3) Heat in B.t.u. carried off by the products of combustion per lb. of carbon when fuel containing hydrogen is burned with an excess of dry air.

$$L'' = \left(\frac{56}{P} + 0.23 + 4.3\ Ha\right) \times (T - t) \quad (3)$$

(4) Heat value in B.t.u. carried off by the products of combustion containing CO per lb. of carbon burned.

$$Lc = 10150 \frac{Pc}{P + Pc} \quad (4)$$

(5) Heat in B.t.u. carried off by the products of combustion due to water vapor in the air supplied in burning a weight of fuel containing 1 lb. of carbon.

$$Lv = Av \left(\frac{117}{P} + 0.88\ Ha\right) \times (T - t) \quad (5)$$

(6) Heat in B.t.u. carried off by the products of combustion per lb. of carbon due to water of hydration in the fuel.

$$Lh - 0.54\ Ov \times (T - t) \quad (6)$$

(7) Heat in B.t.u. carried off by the products of combustion due to moisture contained in the fuel, per lb. of carbon.

$$Lm = Wm\ (0.48\ T + 1076 - t) \quad (7)$$

These formulæ show clearly that the heat wasted up the chimney depends on two factors, the per cent of CO_2 and the temperature of the flue gas. The per cent of CO_2 in the flue gas is an index to the efficiency of the boiler as a heat generator and the temperature is an index to its efficiency as a heat absorber.

Notes on Chemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

The Turin Exhibition.

Your correspondent spent a week in Turin early in September, naturally giving much of the time to the International Exhibition there. There was very much to be seen that was praiseworthy. For one thing the popular sideshows which exasperate the serious visitor were at a minimum.

The general engineering exhibits were decidedly interesting. Of outstanding interest in the frontier between the spheres of engineering and chemistry was the excellent French exhibit of electric furnaces. The American exhibits failed to do justice to American potentialities, and no single section in the United States pavilion was really representative. Metallurgically the British display was worse, and the German display was quite indifferent.

In regard to pure chemistry a special feature of the whole exhibition was the British display, organized by the Board of Trade, of a completely equipped laboratory with apparatus and reagents for both commercial routine testing and for special research. Next to this was a first-class electro-physical laboratory organized by the National Physical Laboratory.

This is not the place to write of the display in such industrial arts as weaving, spinning ceramics or automobiles, where the dispaly was decidedly good. On the electrical side there was a very fine display of heavy generating plant, single-phase railway equipment and extra high-tension overhead material.

The British Industrial Outlook.

Since I wrote my letter at the end of August there has been a decided improvement in the outlook. The Royal Commission on the Railway Strike has reported fully and impartially; the extreme trades union element is, of course, dissatisfied, but the more responsible among their leaders are trying to smooth down the trouble. The South Wales coal area is unsettled, and the neighboring copper works have had trouble. There are, too, still threats of a big coal strike in the north.

Irresponsible socialist extremists are preaching a combined railway and coal strike, which the great mass of the general public, artizan as well as middle-class, will not stand.

The Government efforts to rush through their industrial insurance bill making provision for sickness, and also in certain trades for unemployment, are disturbing to some of their usual supporters. The reason at the root of their action is probably not so much to clear the way for other legislation next year as to discount the force of the Socialist agitation. Disturbed supporters are, however, fearful of the fruits of hurried legislation.

Taking things all round the outlook is better than a month ago, trade conditions are good, and engineering works are now being hampered by that inevitable concomitant of a trade boom, the difficulty of getting the raw materials.

The general confidence is strong, although the European situation still has potentialities of international trouble. Morocco is almost settled, but the Turco-Italian trouble may spread to the Ægean Sea or the Balkans. Assuming the best rather than the worst, economists can then give their attention, and educate an intelligent public attention, to the inter-action of scarcity in food stuffs, heavy gold production and the decreased purchasing power of gold.

Engineering Imports and Exports.

The Board of Trade returns for the nine months ended Sept. 30 give the value of imports of iron and steel ,including manufactures, as £8,201,274, with exports £31,851,601; showing an increase in imports of £1,692,389, and a decrease in exports of £83,303 as compared with the corresponding period of last year. For other metals, including manufactures, the imports were £20,730,879 and exports £8,024,961, the respective increases being £3,136,716 and £398,697.

Imports of electrical goods reached £1,012,643, and exports £1,990,852—a decrease in imports of £142,936 and a decrease in exports of £1,043,873. Imports of machinery were £4,411,648 and exports £22,535,942; showing increases of £1,068,346 and £1,001,425 respectively. Imports of new ships amounted to £60,916, with exports £4,039,482—an increase in imports of £38,581, and a decline in exports of £3,535,579.

The effect of the recent labor troubles is shown by the fact that the returns for the month of September show considerable decreases in the exports of all these classes of goods except non-ferrous metals which increased by £10,810. The decreases are: Iron and steel, £391,337; electrical goods, £222,357; machinery, £350,799; and ships £186,080. Imports of iron and steel increased by £77,025; non-ferrous metals by £435,176, and machinery by £82,331; but imports of electrical goods were lower by £53,532, and the value of imported ships declined £482.

Market Prices in October, 1911.

Copper has been rather dull. Commencing at £55, it had dropped by the 10th to £54-5-0, and, after a short rise, to £56, again dropped and finished £55-5-0.

Tin has been, on the whole, going up. Commencing at £182.0-0 it rose in two jerks to £188-10-0 by the 10th, dropping again to £183-5-0 by the 12th. It then rose to £192 (20th) and closes at £186-15-0.

Lead has been steadily appreciating, with very slight dip on the 10th. Opened at £15-5-0 and closes at £15-15-0.

Hæmatite opened at 61/6, dropped sharply on the 11th, reaching 60/3 on the 12th, then rose to 61/-, and closes at 60/9.

Scotch Pig has been fairly steady, fluctuating only from 52/- to 52/4½. Closes at 52/3.

Cleveland has been moving slightly in sympathy with Scotch Pig. Opened at 46/-, closes at 46/2.

India-rubber has been rather on the decline, opening at 4/7, holding at 4/5½ in mid-month, dropping to 4/3 by the 25th, and closing at 4/3.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	22	0	0
Borax, British Refined Crystal, per ton	16	0	0
Copper Ore, 10 to 25%, per unt.....9/4½ to		9	10½
Copper Sulphate, per ton	20	0	0
Carbolic Acid, liquid, 97/99%, per gal		1	7
Camphor, 1-oz. tablets		1	9½
Caustic Soda, Ash 48%, ordinary, per ton	5	10	0
Hydrochloric Acid, per cwt		5	0
Mica, small slabs, original cases, per lb.....6d. to		2	0
Naphtha solvent, 90%, 160° C., per gal			8

	£	s.	d.
Platinum, per oz., nominal	9	5	0
Petroleum, Russian spot			6
Quicksilver, per flask, Spanish	8	10	0
Sal Ammoniac, lump, first delivery U. K., per ton	42	0	0
Sulphate of Ammonia, f.o.b., Liverpool, per ton	14	5	0
Sulphur, recovered, per ton	5	0	0
Shellac, Standard T. N. Orange Spots, per cwt	4	4	0
Tin Ore, 70% per ton.....£117 to	119	0	0
Zinc, Vielle Montagne	31	2	6

The following differences will be noted:

Rise.	*Amount.* £	s.	d.
Lead	5	5	0
Copper Ore, per unit			3
Quicksilver, per flask		2	6
Sulphate of Ammonia, per ton		7	6
Tin Ore, per ton	5	0	0
Tin, per ton	7	5	0
Copper, Standard, per ton		5	0

Fall.	*Amount.* £	s.	d.
Zinc V. M., per ton	1	7	6
Hæmatite		1	0
Scotch Pig			3
Cleveland			4
Para Rubber			5

Recent Chemical and Metallurgical Patents

Iron and Steel.

Combined Open-Hearth and Electric Furnace for Steel Refining.—The interest taken by the United States Steel Corporation in electric steel refining is further evidenced by a patent of Mr. WM. R. WALKER and Mr. JAMES H. GRAY. Instead of refining steel first in an open-hearth and then transferring it in molten condition to an electric furnace for further treatment, they carry out the whole operation in one and the same furnace, which is first used as open-hearth and then as electric furnace, thus eliminating any transfer of molten metal. (A furnace of this type was in use a year or so ago in Homestead, but none of the results obtained have so far become available for publication.) The furnace has the form of an ordinary open-hearth furnace and is first used as such, silicon, carbon and phosphorus being eliminated by means of an oxidizing slag. This slag is then removed and a fresh basic slag is supplied, while simultaneously electrodes are introduced into the furnace from the top. The electric-furnace treatment now begins and in its non-oxidizing atmosphere the steel is desulphurized and deoxidized. During this second electrical stage of the process, the regenerative chambers are cut off from the main heating chamber. (1,000,217, Aug. 8, 1911.)

Titanium Treatment of Steel.—A patent of Mr. AUGUSTE J. ROSSI, of the Titanium Alloy Manufacturing Company, refers to the treatment of steel with ferro-titanium, only such a quantity of titanium being used as is required to combine with the undesired compounds and elements, and to secure a final product containing substantially no titanium. The object is to procure a close grained, dense structure, without blowholes or cavities. The method is applicable in the crucible, converter and open-hearth steel processes for castings, and in the production of ferro-alloys. (1,003,805, Sept. 19, 1911.)

Electrolytic Refining of Iron.—For the electrolytic production of pure iron, free from carbon, from impure iron, like cast or wrought iron scrap, cast iron turnings or borings, etc., Mr. ALEXANDER S. RAMAGE, of Buffalo, N. Y., uses the process sketched in Fig. 1. The electrolytic cell 1 is divided by a diaphragm 2 of asbestos cloth into anode and cathode compartments 3 and 4. The anodes 5 are of sheet lead, the cathodes 6 of iron. When a solution of ferrous sulphate is electrolyzed in both compartments, iron is deposited on the cathodes, while ferric sulphate and sulphuric acid are formed at the anode. From the anode compartment the acid liquor is passed into the scrubber 7, where it is subjected to the action of a small proportion of sulphur dioxide. (Only enough sulphur dioxide

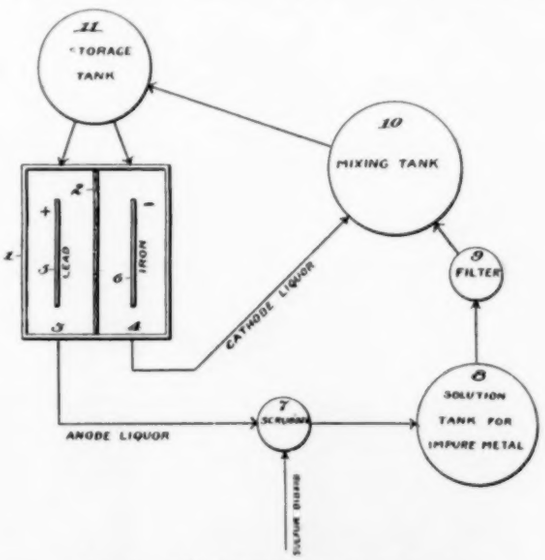


FIG. 1.—ELECTROLYTIC REFINING OF IRON

is used to form sufficient sulphuric acid to replace that lost by leakage, but not so much as to effect the reduction of the ferric sulphate to ferrous sulphate.) The acid liquor then passes to solution tank 8, where it is brought in contact with the impure iron to be refined (scrap, etc.), together with ferrous sulphide, which under the action of the acid liquor generates hydrogen sulphide. Under these conditions there occurs a complete pre-

cipitation of those metals whose sulphides are insoluble in dilute acids. To aid the solution of the iron, heat is employed, and the solution tank is also fitted with an insoluble anode buried in the scrap and with an iron cathode in the liquor above the scrap, so as to aid in the solution of the scrap iron and diffuse the hydrogen throughout the solution. The hydrogen and the sulphuretted hydrogen resulting from the solution of the ferrous sulphide effect the reduction of the ferric sulphate to ferrous sulphate. When the liquor has become practically neutral, it is forced through a filter press 9, flowing thence to a mixing tank 10, where it is mixed with the effluent liquor from the cathode compartments. The mixed liquor flows to a storage tank 11, whence it is distributed to the several anode and cathode compartments of the electrolytic cells, thus completing the cycle of operations. The preferred temperature during electrolysis is 145 deg. Fahr., a suitable current density is 10 amp. per square foot, the voltage approximating 2.5 volts per cell. With revolving cathodes current densities approximating 100 amp. per square foot may be used, with a correspondingly increased voltage. The chief feature of the process is that it avoids the solution of such metals as are liable to contaminate the electro-deposited iron, notably carbon and certain metals, such as arsenic and antimony, whose sulphides are insoluble in dilute acid solutions. (1,007,388, Oct. 31, 1911.)

Steel for Armorplates.—A low-carbon steel for armor plates and gun tubes is patented by Mr. Wm. Beardmore, of Glasgow, Scotland. In addition to the usual ferrite, it contains a percentage of carbon, not exceeding 0.25, and molybdenum and nickel in proportions sufficient to compensate for the deficiency in the strength and toughness of the steel due to the low percentage of carbon. The composition is 0.15 to 0.25 per cent C, 1 to 2 per cent Mo, 5 to 7 per cent Ni, further not over 3 per cent W, and not over 0.7 per cent Mn. This steel does not sufficiently harden by mere quenching when hot. It must be subjected to a process of cementation. By suitably regulating the degree and extent of differential hardening treatment to which the steel plates are subjected, it is possible to secure great hardness in some parts while retaining at other parts a ductile and tough condition. "In the case of armor plates the qualities possessed by the steel enable them after cementation to resist the penetration of modern capped projectiles and at the same time to resist the racking action of the projectiles, particularly in the case of plates of less thickness than the caliber of the guns used in attacking them. In the case of steel for gun tubes, an increased resistance to bursting or cracking of the tube by the shock of discharge is obtained. (1,007,055, Oct. 31, 1911.)

Treatment of Iron and Steel to Prevent Rusting.—Mr. Thomas Watts Coslett, of Birmingham, England, proposes to prevent the rusting of iron or steel by subjecting it to an acid solution of zinc phosphate. A concentrated solution of 6 oz. zinc, 1 pint phosphoric acid and 1 pint water is made up and applied directly to the iron or steel articles for a short time or is diluted with water before the iron or steel articles are immersed in it. For dilution 1 oz. of the concentrated solution is added to each gallon of water. For the purpose of renewing the dilute solution there is added to each gallon thereof ¼ oz. or less of the concentrated solution as and when required. Under certain circumstances it is of advantage to treat the iron with a solution containing both zinc phosphate and iron phosphate; in this case iron filings and granulated zinc are dissolved in phosphoric acid. "The formation of the deposit on the iron or steel surface under treatment may be accelerated by passing an electric current through the dilute solution, which may be either in a hot or a cold condition," with iron anodes. (1,007,069, Oct. 31, 1911.)

Gold and Silver.

Continuous Vacuum Filter.—A novel form of continuous filter for separating cyanide solution from ore pulp has been patented by Mr. Bruce W. Taylor, of New York. As shown in Fig. 2, which is a sectional elevation, the machine consists of a tank in which a chain of filtering pans is mounted, adapted to travel around sprocket wheels, one of which is mounted outside and slightly above the top of the tank, while the other is located in the lower part. The vacuum pans have the usual construction of wire cloth, cocoa matting and filter cloth, and each pan has suitable connections with vacuum and

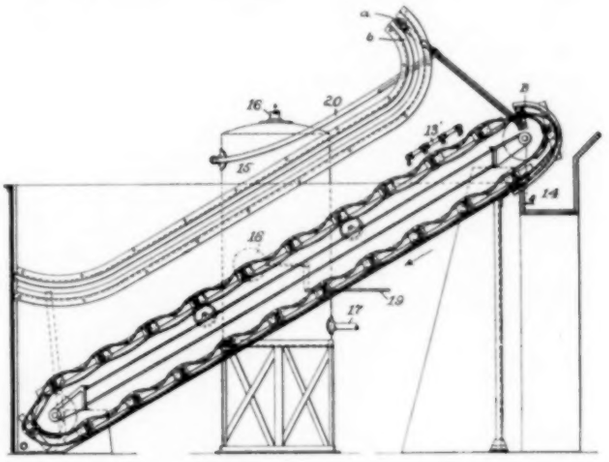


FIG. 2.—CONTINUOUS FILTER

air pressure. The point at which the pans are connected with the vacuum and where the air pressure closes is indicated at *A*, while the point at which vacuum is cut off and air pressure applied is shown at *B*. As the pans descend into the tank which is filled with the pulp to be filtered, they are connected with vacuum and consequently accumulate a cake of slime. As the pans rise out of the tank they first pass under a water spray shown at 13, where they receive a wash. Later the vacuum is broken at *B* and air pressure applied which forces the cake from the pan into a discharge launder 14.

The vacuum tank is indicated at 15, and has dry vacuum connection at 16 and wet vacuum connection at 17. The pump operating the latter is controlled by a float 18, 19, so that it will work fast or slow according to the amount of solution in the tank. There is a continuous vacuum on the hose 20,

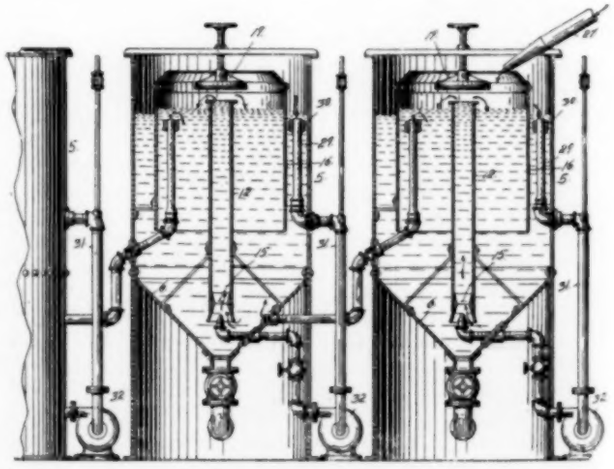


FIG. 3.—CONTINUOUS AGITATOR

and the vacuum on the several pans is controlled by suitable valves. The pipe connection at the end of the hose is provided with a trolley *a* and track *b*, and the pipe travels back and forth as the chain of filter pans revolves, the hose also following the line of travel. (1,000,856, Aug. 15, 1911.)

Pulp Agitator.—The improvement made by continuous agitation of ore pulp in cyanide solution has led to a number

of designs of agitators being patented. One of the latest is that of Mr. WILLIAM C. PATERSON, of Denver, Col. The agitator may be used either singly or in plurality. In the accompanying drawing, Fig. 3, the agitator is shown in plurality for continuous work.

The machine belongs to that class of agitators in which a centrifugal pump is the means of agitating the pulp, and in which means are provided in the tank for permitting only clear solution to go to the pump, thereby avoiding excessive wear due to the abrasive action of the mineral particles. In the drawing the number 5 designates a cylindrical tank having a cone bottom 6, with a central pipe 12, which is slightly reduced in diameter at 15. Surrounding the upper part of the central pipe is a circular partition 16, while immediately above the central pipe is a baffle plate 19, which is movable vertically. Pulp is fed to the system at 27, and the agitation and circulation are accomplished by centrifugal pumps 32. The pump draws practically clear solution from outside the circular partition, through the pipe 31, which terminates in a flexible portion 29, with a float 30.

Similar arrangement is provided on the opposite side of the tank, but within the circular partition, for the purpose of carrying a portion of the pulp from one tank to another. By this arrangement, agitation is accomplished by the circulation of clear solution through the central pipe, and the transfer of pulp through the system is made by continuously drawing off a uniform sample of the pulp as it falls from the top of the central tube. (1,000,689, Aug. 15, 1911.)

Improvement in Zinc Dust Precipitation.—Mr. CHARLES W. MERRILL, of Berkeley, Cal., who had used zinc dust precipitation of cyanide solutions for some years, has recently been granted letters patent for an improvement in the system. He claims that the ordinary zinc fume or dust is rendered less efficient as a precipitant on account of the slight coating caused by oxidation of the zinc. It is further claimed that this inefficiency is more noticeable in treating low grade or weak solutions. Mr. Merrill proposes to add to the usual precipitating system a grinder, such as a small tube or barrel mill, through which the entire mixture of gold-cyanide solution and zinc dust is to be passed. This apparatus will be placed somewhere between the tank containing unprecipitated solution and the filter press which is to separate the solid from the precipitated solution. The grinding operation will remove the deterrent coating from the zinc fume and increase its efficiency as a precipitant. (1,006,865-6, Oct. 24, 1911.)

Copper.

Copper Converter Tuyere.—Another patent has been added to the list of those issued to Mr. ELIAS A. C. SMITH, of Baltimore, Md., in connection with the development by him and Mr. Peirce of the basic copper converter. This latest patent refers to the construction of the converter tuyere, and the only claim reads as follows: "A blast tuyere, provided at its outer end with a Dyblie valve and its casing having a removable cap, the main bore of the casing being in alignment with and of the same diameter as the bore of the tuyere, and said cap having a perforation of but slightly less diameter than that of the tuyere; whereby, during the continuance of the blast, a punch bar may be employed, making a machine fit with said perforation and approximately but loosely fitting the bore of the tuyere, and whereby, on removing the cap during interruption of the blow, a second punch bar may be employed, still more neatly fitting the bore of the tuyeer." (1,007,149, Oct. 31, 1911.)

Treatment of Copper-Zinc Sulphides in a Converter.—With a view to improving the methods used for the treatment of copper and zinc sulphide ores, M. DÉSIRÉ KORDA, of Paris, has invented a procedure which he claims to be more economical than any yet devised. He provides an ordinary converter of the kind used for converting copper and uses a high proportion of coke in its lining. This lining may be formed of an intimate mixture of 60 parts quartz, 20 parts coke and 20 parts refractory clay. The lining around the tuyeres has no coke. After the converter is dried and heated in the usual manner and lined with incandescent coke, covering the tuyeres with combustible material. A few minutes' blowing brings the contents to a white heat, when the ore is then charged until the converter is full. It is preferable to charge the ore in the form of rock. In order to start the operation quickly, the cleanest sulphides are charged first, the less fusible ores following. Finally a charge of coke of nut size is added.

The blast is continued at a moderate pressure until a liquid bath forms in front of the tuyeres, after which the pressure may be increased. At the end of an hour the charge is melted to a homogeneous mass and the maximum blast is obtained. Zinc is reduced and discharged at the mouth of the converter, where it burns to oxide and is conducted to recovery chambers. Copper is converted to matte, and the operation is stopped when the latter approximates 60 per cent copper. Five per cent of pyrite is now added to the converter, having the effect of cleaning the slag of copper. The converter is then skimmed and the matte blown to metal.

The following figures are given by the inventor as showing an actual test of the process: Ore: Cu, 677 per cent; Zn, 16.31 per cent; Fe, 24.00 per cent; S, 23.00 per cent; SiO_2, 15.00 per cent. Matte: Cu, 67.22 per cent; Zn, 2.36 per cent. Slag: Cu, 0.54 per cent; Zn, 4.64 per cent. Oxide: Cu, 0.10 per cent; Zn 55.00 per cent. The net recovery is thus 93 per cent of the copper and 78 per cent of the zinc. Owing to slight waste in the recovery of the latter, however, this figure is reduced to 60-70 per cent of the zinc really recovered in the form of oxide. (1,003,051, Sept. 12, 1911.)

Lead and Zinc.

Refining Lead and Lead Alloys.—The removal of impurities from lead and the alloys of lead with antimony and tin is highly desirable from a commercial point of view. Chief among these impurities is arsenic, the presence of which renders lead unfit for certain commercial uses. Mr. LELAND E. WEMPLE, of St. Louis, Mo., proposes to accomplish the removal of arsenic and other impurities by treating the molten lead or lead alloy with an alkaline hydroxide, such as sodium hydroxide, bringing the metal in intimate contact with the solvent for the impurities.

The apparatus used is represented diagrammatically in Fig. 4. It consists of a series of pots adapted to be heated from fire boxes, and having spouts for conducting molten metal from

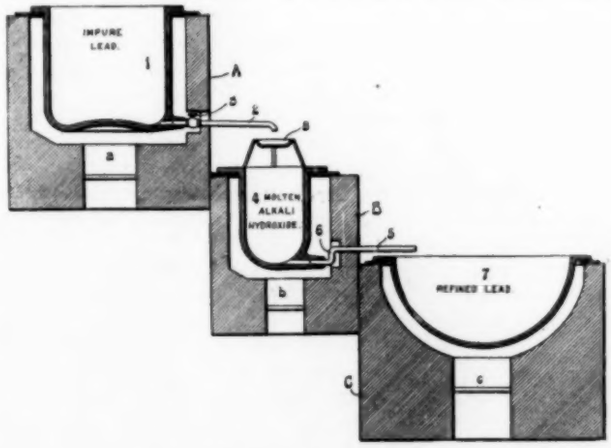


FIG. 4.—LEAD REFINING

one to another. The impure metal is contained in pot 1, and is conducted to the alkali pot by pipe 2, falling first into a perforated pan 8 placed above the alkali pot. This has the effect of disintegrating the molten stream into innumerable globules, exposing a large surface of metal to the action of the alkali. The refined metal falls to the bottom of the alkali pot and passes through the pipe and trap 5, 6, into the pot 7. It will be

inevitable that some of the alkali will also pass into pot 7, and this is removed by treating the molten mass with dry nitre cake, which is forced to the bottom of the pot 7. This quickly unites with any alkali in the mass and rises to the surface, forming a slag which is skimmed off.

If it is desired to refine lead alloys, such as those of tin and antimony with lead, it is necessary to retard the action of the alkali, so that tin and antimony will not be dissolved. This is done by adding a reducing agent such as zinc dust, iron filings or aluminium powder. In case iron scale is used, the inventor prefers to mix 20 parts of iron scale with 100 parts of alkali hydroxide in the reaction pot.

For the purpose of accelerating the removal of some impurities which are not readily acted upon, such as bismuth, tin and antimony, it is well to mix with the alkali hydroxide an oxidizing substance such as the nitrate, chlorate or peroxide of sodium or potassium. (1,006,323, Oct. 17, 1911.)

Extraction and Electrolytic Precipitation of Zinc.—The treatment of low grade zinc ores, containing from 5 to 15 per cent of zinc, with or without other metals, is a problem which attracts a great deal of attention from metallurgists. In a patent recently granted to Messrs. John R. Williams, Henry W. Bradley and Benjamin Bradley, of Sheffield, England, a process is outlined which contemplates the removal of the zinc by dissolution in a weak acid solution of zinc sulphate, and the electrolytic reduction of zinc from this solution.

Sulphide ores must be first roasted, after which they are leached in a suitable tank with a solution containing 6 to 8 per cent neutral zinc sulphate and 0.3 per cent sulphuric acid. These strengths were arrived at by experimentation. It was found that with quantities of zinc sulphate from 5 to 16 per cent the rate of deposition was no greater than at 5 per cent, while below 5 per cent the rate rapidly decreased. Also, with the higher percentages of zinc sulphate in the extracting solution, the loss of solution in the ore was too great. The acid strength is determined by the tendency of the ore to form a gelatinous mass with the acid, and by the effect of the acid on the cathode. With regard to gelatinization, the limit may be 2 per cent, but with regard to the action on the cathode the limit must be as low as 0.5 per cent. If the current density does not exceed 11 amp per square foot of lead anode surface, the anodes may be run for 300 days continuously without renewal. It is preferable to use a current density of about 10 amp per square foot of surface, and to start electrolysis with a neutral solution.

The inventors have a preferred form of cell which they use for electrolysis. It consists really of a number of practically separate cells parallel to one another. The cathodes are made of iron, preferably tempered and galvanized, and the anodes of lead plate. The anodes are placed one on either side of the cathode so that both surfaces of the latter are operative.

Copper ores may be similarly treated with a slightly acid solution of copper sulphate, or mixed copper and zinc ores may be leached out together and recovered separately by electrodeposition under proper electrical conditions. (1,006,330, Oct. 17, 1911.)

Electric Zinc Furnace.—Two patents of Mr. A. L. J. Queneau, of Philadelphia, Pa., refer to an electric zinc furnace which, in its latest construction, is shown in Fig. 5. It is a rotary furnace provided with a molten resistor D of molten iron containing 12 per cent phosphorus (to increase the electric resistivity) or preferably of molten slag. To get high fluidity of the slag, florspar or phosphoretic slag from an open-hearth steel furnace may be added. The furnace has an outer metallic shell *k* of low-carbon steel. At the feed end it is closed by a metallic plate *h*, insulated from the metallic shell *k* and provided with a flange *v* for making contact with one of the brushes *y* of the external electric circuit. The opposite end of the furnace is closed by a plate h^1 which is electrically connected with the metal shell *k*. The other brush *Z* of the external elective circuit makes contact with the shell *k* through the intermediary of the tire ring *d*. The shell *k* is provided with an outer lining of fire brick and an inner lining of magnesite brick or chrome brick. The end courses *C* are made of a conducting mixture of graphite, dead-burned magnesia, and tar. Exterior to these courses *C* are the raphite end linings *n*, adjacent to the end plates *h* and h^1. The electric circuit is, therefore, completed from brush *y* to plate *h*, to grpahite end lining *n* and conducting course *C* to the fluid resistor *D*, thence to the conducting course *C* at the other end of the chamber, the graphite lining *n*, plate h^1, shell *k* and ring *d* to the other brush *Z*.

At the left hand of the diagram the condenser is shown, with the shell k^1 lined with graphite compounded with clay Within the condenser are arranged the baffle dip partitions *L*, provided with apertures *o*. The construction may best be seen from the separate diagram *A* shown at the top left corner of

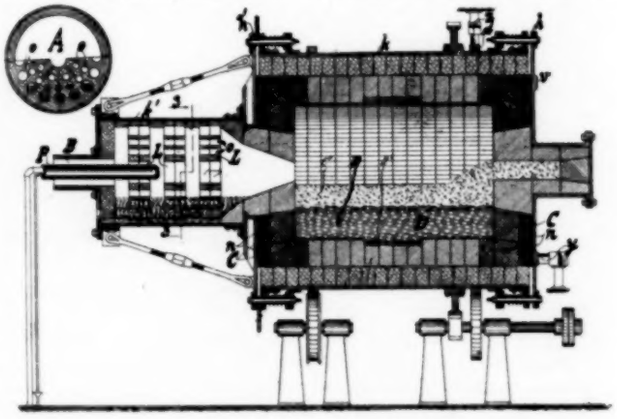


FIG. 5.—ELECTRIC ZINC FURNACE

the illustration, which gives a cross section of the condenser along the lines 33 in the main diagram. The combustible gases, such as carbon monoxide, escape from the condenser through the prolong *B*. To regulate the temperature in the condenser, the pipe *P* is provided through which cooling water is passed.

The charge consists of a mixture of zinc ore and coke, wherein the coke is but slightly in excess of that theoretically required for reduction of the ore. The charge is preheated to as high a temperature as the coke will stand without ignition, say 500° C, within a rotary furnace of the muffled type, heated from the outside and containing a number of steel balls so as to serve as a ball mill. The preheated charge is then supplied to the electric furnace. (1,006,876 and 1,006,877, Oct. 24, 1911.)

Nickel.

Treatment of Nickel-Copper Ores.—An improvement in the treatment of the nickel-copper ores of Sudbury, Ontario, Canada, is made the basis of a patent granted to Franklin R. Carpenter, deceased, and Arthur H. Carpenter, his son, of Denver, Col. The process has to do with the removal of the copper and sulphur from the ore, leaving the iron and nickel as a roasted product suitable for reduction to ferro-nickel alloy. The inventors first give the ore a dead roast, leaving the iron, copper and nickel practically as oxides. They then add a suitable amount of raw pyrite or pyrrhotite, together with the necessary salt, and give a chloridizing roast. Owing to the greater ease with which copper is chloridized, the process can be controlled in this respect, so that when the resulting mass is lixiviated only copper will be leached and nickel will remain with the iron. The nickel iron residue can then be smelted directly or with other iron ores for the production of ferro-nickel alloys, thereby saving the cost of separating the nickel and giving complete utilization of the iron. (1,006,355, Oct. 17, 1911.)

Alloys.

Ferrochromium Made in Continuous Resistance Furnace.—Instead of a discontinuously operating arc furnace, Mr.

EDGAR F. PRICE, of the Electro Metallurgical Company, proposes to use a continuous resistance furnace for the production of ferrochromium. A body of molten metal occupies the lower portion of the furnace. Above this rests a deep body of molten slag, and on the top of it is placed the charge mixture of chromite, coke, lime, etc. The body of slag is heated by a current passed through it between electrodes suspended from the top into the charge mixture and into the layer of molten slag. As reduction proceeds, the iron and chromium alloy and the alloy settle through the slag into the layer of metal beneath. The slag produced combines with the slag resistor and the excess slag and metal are removed from time to time through tap-holes, additions being made to the charge as required. The reduced metals thus quickly separate from and are maintained out of contact with the free carbon in the charge. The resulting alloy is, therefore, low in carbon. (995,481, June 20, 1911.)

Nickel Titanium.—To produce a binary alloy of nickel and titanium, Mr. AUGUSTE J. ROSSI, of the Titanium Alloy Manufacturing Company, mixes nickel, nickel oxide and titanic acid with sufficient aluminium for the reduction of the oxides. The mixture is then heated in an electric furnace and the reduced titanium *in statu nascendi* alloys with the nickel. The reactions during the reductions are

$$3\ TiO_2 + 4\ Al + Ni = 2\ Al_2O_3 + Ni + 3\ Ti$$

and

$$3\ NiO + 2\ Al = Al_2O_3 + 3\ Ni$$

Instead of aluminium, carbon may be used as a reducing agent, but a higher temperature is then essential. (1,003,806, Sept. 19, 1911.)

Aluminium Bronze.—Mr. P. H. G. DURVILLE, of Paris, France, patents the use of manganese as a purifier for copper-aluminium alloys. He uses the following composition: 86 to 95 per cent copper, 0.05 to 5 per cent manganese and 6 to 11 per cent aluminium. The metals can be melted in the order indicated, but a portion at least of the manganese must be added in the form of an alloy with copper or with aluminium at the time of casting. Manganese in small proportions plays the part of a purifier as regards impurities other than oxides, which are not removed from the alloy by aluminium. The greatest proportion of manganese, up to 5 per cent, increases the hardness and the malleability of the alloy. These alloys have all the qualities of binary aluminium bronzes (consisting of copper and aluminium only), that is, great strength, good color and resistance to chemical agents, while being very easy to work. They are characterized moreover by the advantage that very sound ingots are obtained which can be easily forged, stamped, rolled or drawn. (1,007,548, Oct. 31, 1911.)

Pyrophoric Metal Alloy.

An improvement in pyrophoric bodies is claimed in a patent granted to GEORG F. HOFMANN, of Munich, Germany. It consists in adding metallic cerium to an alloy of manganese and antimony. Pyrophoric bodies of antimony and manganese are known, but their properties are not such as adapt them for use in ignition appliances, for the reason that they are not easily abraded by files or other frictional means. The inventor claims that the addition of about 5 per cent metallic cerium renders the antimony manganese alloy soft enough to be easily filed or abraded by friction so that particles fly from the body in a stat of incandescence. This alloy is different from that in which cerium is the main source of the pyrophoric property, where about 70 per cent cerium and 30 per cent iron are combined. (1,006,849, Oct. 24, 1911.)

Chemical Engineering.

Generating Steam from Molten Slag.—In Figs. 6 and 7 are shown two recent methods of generating steam from molten slag. Fig. 6 shows the arrangement designed by WALTER L. JOHNSON, of Middlesbrough, England. The main points in his invention comprise means for eliminating objectionable gases from the steam before use. Referring to the figure, *a* represents a slag spout which delivers hot slag from a furnace to the boiler *c*. A stirrer *d* may be used if desired. The boiler contains water and milk of lime for the absorption of sulphurous gases. The granulated slag is removed by elevators as shown, while the steam passes through the tower *h* to the tower *i* where it encounters a falling spray of hot caustic soda contained in the tank *k*. This is designed to neutralize any acid

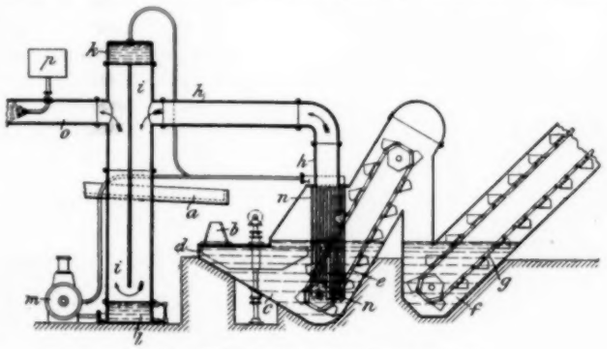


FIG. 6.—GENERATING STEAM FROM SLAG

gases, but a further neutralization is provided for by an ammonia spray *p* in the pipe *o*. The caustic solution is circulated by the pump *m* through coils *n* in the boiler, so that it is kept at a temperature of about 212 deg. Fahr. The ammonia may be recovered by condensing the engine exhaust. (1,005,711, Oct. 10, 1911.)

Fig. 7 represents the design of WILLIAM E. MCKEE, of Bisbee, Ariz. It compries a boiler and elongated leg in which is a bucket elevator for the removal of the granulated slag. The molten slag is first received in a hopper placed above the boiler. Between the hopper and boiler is a specially designed valve which can be closed to shut off the flow of slag and prevent the escape of steam. An opening for the escape of steam to the

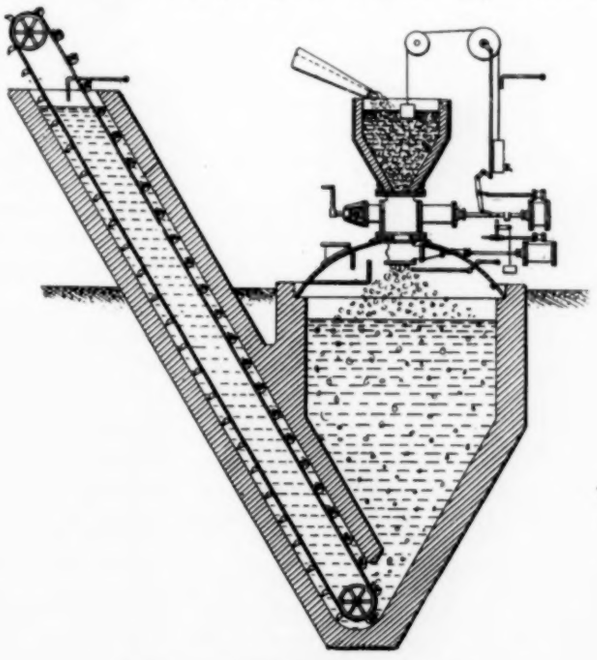


FIG. 7.—GENERATING STEAM FROM SLAG

point of utilization is shown in the boiler head. It will be noticed that provision is made for a difference in head of the water in the boiler proper and in the leg. This difference amounts to 2 ft. head for every pound of steam pressure in the generator. Feed water is introduced through the leg so that practically all the heat of the slag is uilized before its discharge. (1,001,290, Aug. 22, 1911.)

Electric Furnace Designs.

Electric Resistance Furnace.—In electric resistance furnaces with an electrode at the top and an electrode in the bottom, the upper electrode is kept in good contact with the upper part of the charge. When the charge sinks down in the furnace, this contact is liable to be broken and an arc is formed. This increases the resistance and reduces the power supply to the furnace and causes other disadvantages, like greater electrode consumption, concentration of the heat on the top, etc. To prevent this, Mr. ALBERT PETERSSON, of Odda, Norway, arranges the feeding device in such a way that the fresh charge is fed in a horizontal direction toward the upper electrode pressing against it and preventing the formation of arcs. Quite a number of such arrangements are described, of which one is shown in Fig. 8 (1,002,444, Sept. 5, 1911).

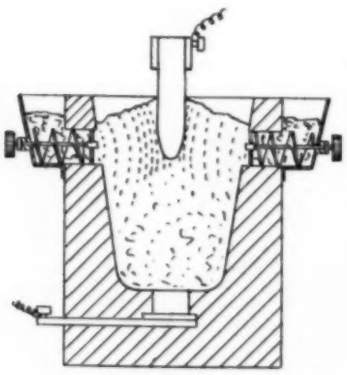


FIG. 8.—ELECTRIC RESISTANCE FURNACE

Electric Furnace with Conducting Hearth.—Mr. GEORGES MASSIP, of Paris, France, patents an electric furnace with an upper movable electrode, while the hearth itself acts as the lower electrode. The hearth is made of a material, like magnesia, which is a non-conductor of electricity when cold, but becomes a conductor when heated. To heat it, there are embedded within the hearth (but removed from the inner surface of the hearth) metallic heating resistors, suitably connected, like the upper electrode, to an electric circuit. (1,005,725, Oct. 10, 1911.)

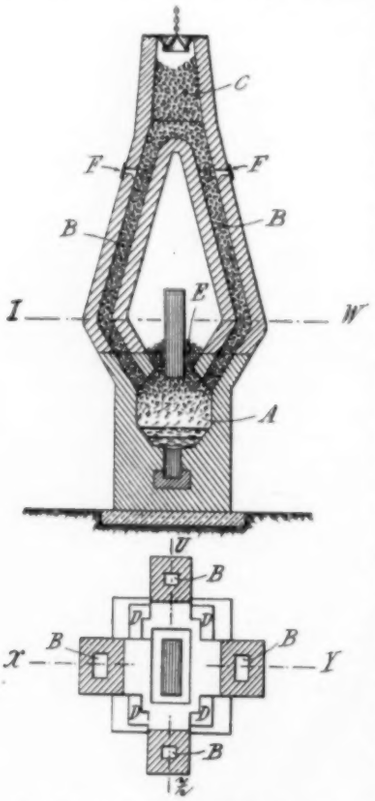


FIG. 9.—ELECTRIC REDUCTION FURNACE

Electric Reduction Furnace.—Mr. HANS BIE LOVENTZEN, of Notodden, Norway, has patented the furnace shown in vertical and horizontal cross-section in Fig. 9 for operation with single-phase current by means of an upper and a lower electrode. Surrounding the upper electrode are a series of vertical channels *B* which terminate in a common charging shaft *C*. By means of a fan, the gaseous products of reaction are withdrawn through the channels. Through the valves *F* air may be blown into the channels, if desired. The furnace construction is such as to permit separate feeding of ore and carbon, namely, the ore *C* through the channels *B* and the coke *E* around the upper electrode. (996,492, June 27, 1911.)

Electric Furnace.—An electric-furnace construction of Mr. ALVOIS HELFENSTEIN, of Vienna, Austria, is shown in Fig. 10, in vertical and horizontal cross-section, the two diagrams not being drawn to the same scale. The two side walls *A* and *B* are connected by means of the end walls *g* and *h*. On the top of the furnace two parallel supports d_1 and d_2 are arranged in the form of water-cooled girders, resting on the end walls of *g* and *h*. The two girders are spaced apart so as to form a charge opening, which extends the whole length of the furnace chamber, and a superposed receptacle *b* is provided to confine the charge. The lateral arched coverings of the furnace chamber a_1 and a_2 are likewise supported by those girders. An upper vertical electrode *i*, longitudinally arranged, is disposed in the

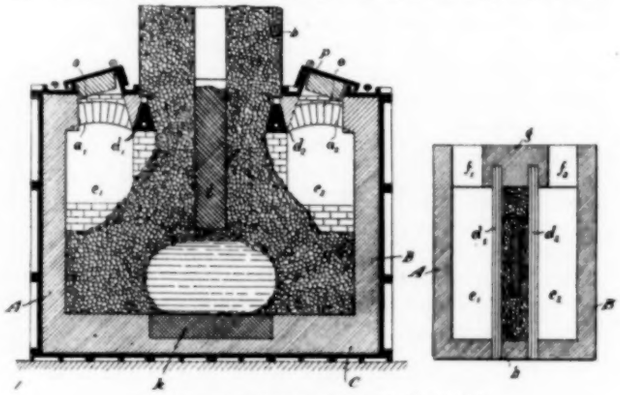


FIG. 10.—ELECTRIC REDUCTION FURNACE

charge opening and extends nearly the whole length of the furnace chamber. Horizontally arranged in the bottom *C* of the furnace is the bottom electrode *k*, also extending nearly the whole length of the furnace chamber. The openings *o* are provided for inspection and manipulation of the charge. As will be seen, on both sides of the furnace, above the charge, separate chambers e_1 and e_2 are formed into which the reduction gases can escape, being subsequently withdrawn through the outlets f_1 and f_2. (1,000,805, August 15, 1911.)

In another patent of Mr. Helfenstein a slightly modified furnace construction is shown. The top of the charging shaft is closed and the whole arrangement is again such that free spaces are formed on the sides above the charge, into which the gases evolved during the reaction can escape. If they are combustible it is proposed to utilize their calorific effect within the furnace. For this purpose air is blown into these spaces and the gases are burnt, thus preheating the ore before it enters the melting zone. (1,002,988, Sept. 12, 1911.)

A third patent of Mr. Helfenstein refers to the construction of a battery of electric furnaces, all with one single common bottom electrode. That is, a single massive body serves as a common bottom to all of the furnaces and serves as bottom electrode for all of them. Upon this bottom various furnace shafts are constructed, whereby each single shaft may be used as an independent working hearth so that it is possible to conduct melting operations of different character in the different furnaces. (1,002,843, Sept. 12, 1911.)

Two-Phase Electric Furnace.—A patent of Mr. AXEL R. LINDBLAD, of Ludvika, Sweden, refers to a two-phase furnace,

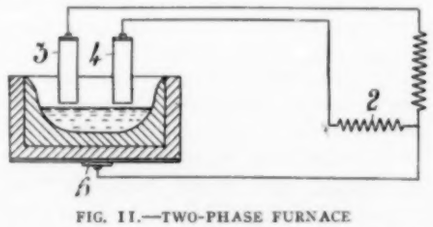


FIG. 11.—TWO-PHASE FURNACE

which has already been the subject of an article by Mr. T. D. Robertson in our November issue (p. 573). The essential electrical connections are shown in Fig. 11. 1 and 2 are the

two phases of a two-phase system, each being connected with one of the upper electrodes 3 and 4, while the neutral point is connected with the contact 6 in the bottom of the furnace. As the two circuits have a phase difference of 90 deg. relatively to each other, the current which passes through the smelting bath and contact 6 is V_2 times the current passing through each of the electrodes 3 and 4. The tension between both electrodes is also V_2 times the tension consumed in each arc. If the current supply is three-phase, it is changed to two-phase by the well-known Scott transformation. (1,002,286, Sept. 5, 1911.)

Electric Furnace.—Mr. LARS YNGSTRÖM, of the Stova Kopparbergs Bergslage Aktiebolaz, of Falun, Sweden, patents the furnace shown in Fig. 12. A crucible 1 forms the working or melting chamber, and above it is provided a stack 2 for feeding the charge downward. But between the walls of the stack and the crucible there is a ring-shaped space 4 through which more charge is introduced into the furnace, while it also serves for the introduction of the electrodes 3, which are thus surrounded by charge material. (Bottom electrodes may also be provided.) A vault or roof for the crucible which would be liable to destruction is thereby avoided. 6 is the tapping hole. Two modifications of this furnace design are also described. (1,001,915, Aug. 29, 1911.)

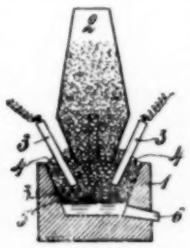


FIG. 12—ELECTRIC REDUCTION FURNACE.

Furnace Electrodes.

Dimensioning of Furnace Electrodes.—Mr. CARL HERING'S well-known laws of electrode losses are the basis of a recent patent granted to him for dimensioning electrodes. He proposes to dimension them in such a way that no heat passes through the hot electrode end (within the furnace) in either direction. No heat flows, therefore, from the furnace interior through the electrode to the outside. Only the heat generated within the electrode from electrical energy (the C^2R heat) passes out of the cold electrode and to the outside. The first claim reads as follows: "In an electric furnace, an electrode having a resistance such that the C^2R heat developed therein by current transmitted therethrough to said furnace shall be substantially equal to twice the heat conduction loss when no current flows." (999,719, Aug. 1, 1911.)

A second patent of Mr. Hering relates to this dimensioning of electrodes in combination with other features. It will suffice to give here the first claim: "In an electric furnace, molten conducting material serving as a furnace resistor, and an electrode in electrical communication with said resistor for communicating heating current to said resistor, the length and cross-section of said electrode being so related that the resistor heating current passed through said electrode raises said electrode to a temperature preventing heat conduction through said electrode from said molten material." (999,720, Aug. 1, 1911.)

Protective Coating for Carbon Electrodes.—To protect carbon electrodes against oxidizing influences, Mr. JOS. L. K. SNYDER, of the National Carbon Company, of Cleveland, Ohio, provides a protective coating to the electrode surface, while in a green (unbaked) and therefore plastic condition. During baking or later in the electric furnace this coating is automatically changed by the heat into a protective and very firmly adhesive slag. If the carbon mix, of which the green electrode is formed, contains acid impurities, like silicious material, sodium carbonate with a binder may be used for the coating. Under the influence of heat this will combine with the silicious material and form a slag-producing glazer. Of, if the carbon mix contains basic impurities, like ferric oxide, potassium bisulphate or borax with a binder may be used for the protective coating. Or calcium chloride may be used for the coating, as it will form with the carbon calcium carbide. Pitch and tar are suitable binders. (1,000,761, Aug. 15, 1911.)

Large Electrode.—The construction of an electrode for very large currents, like 25,000 amp., for calcium carbide furnaces is the subject of a patent of Messrs. WM. T. GIBBS and R. A. WITHERSPOON, of the Shawinigan Carbide Company. The head of the electrode (Fig. 13) consists of a casting of circular shallow horizontal box form *b* with a diametrical hollow flange *c* upon its top. The electrode head is water-cooled, *i* and *j* being the feed and discharge pipes for the box and *k* and *l* for the flange. The flange *c* acts as a terminal and also as a means for connecting the electrode with a crane. The horizontal box position *b* has a central opening encircled by a series of similar openings, all provided with screw-threaded sockets. A series of short graphite plugs *m* are screwed into the sockets in the head, and into their lower ends (as at *o*) are threaded amorphous carbon rods *p*. Thus these electrode pencils consist in their upper parts of graphite and in their lower parts of amorphous carbon. A sheet-iron shell encloses the pencils and a filler *s*, consisting of 100 parts by weight of ground coke, 27½ parts redistilled tar and 27½ parts pitch, is packed around and between the pencils, and is preferably molded and baked before the shell is set in place. The reason for using a graphite electrode in the water-cooled head is that amorphous carbon, when used in such water-cooled heads, is liable to break, while graphite does not break. For the lower parts of the pencils amorphous carbon is used on account of its cheapness. The electrode is shown in the illustration applied to a calcium carbide furnace, supported on a wheel truck 4 and having a carbon base 2 and limestone walls 3. (1,003,354, Sept. 12, 1911.)

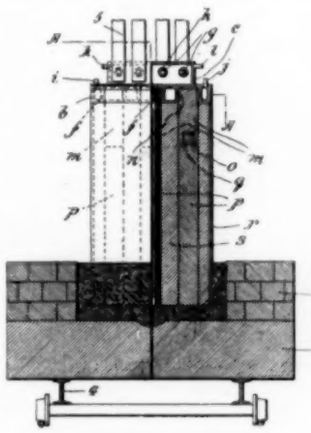


FIG. 13—LARGE ELECTRODE OF GRAPHITE AND CARBON.

Various Electric Furnace Processes.

(See also above under the headings for the different metals.)

Electrically Heating Carbon.—In electrically heating carbon, for the production of electrode compositions, battery fillers, etc., a close adjustment of time and temperature is required to convert the given raw material into a product of the desired character. While in practice the intermittently operated furnace is always employed, Mr. WILLIAM ACHESON SMITH, of the International Acheson Graphite Company, has devised a continuously operating furnace in which the charge is progressively advanced in a direction parallel to the lines of flow of the electric current. In this way the charge may be subjected for any desired period (depending upon the rate of advance of the charge) to uniform temperature conditions. Two slightly different designs of the furnace are shown in Fig. 14. In the left-hand drawing the furnace charge 8 serves as the resistor interposed between the upper and lower electrodes 6 and 7. The lower electrode is supported by a hollow water-cooled metal bar or casing 9 carried by the furnace walls. In the furnace walls adjacent the heating zone a hollow space 4 is provided which communicates by inclined apertures 4*a* both with the heating chamber and the external air so that the furnace gases may readily escape from the heating zone and a mantle of such gases may be burned within the furnace wall to economize heat. Water pipes 5 are provided between the heating zone and the discharge hopper. The latter is provided with a slightly coned disk, rotated by a worm gear, so that the material is discharged over the periphery of this disk only while it is rotating. In the right-hand diagram the upper and lower terminals 6 and 7 are connected together by a carbon or graphite rod 14, which is particularly useful when the charge,

in the cold state, is a poor conductor of electricity. There are also some other small differences in design in the two furnaces, which can be seen in the illustration. (Process patent 1,004,923; furnace patent 1,004,924, both of Oct. 3, 1911.)

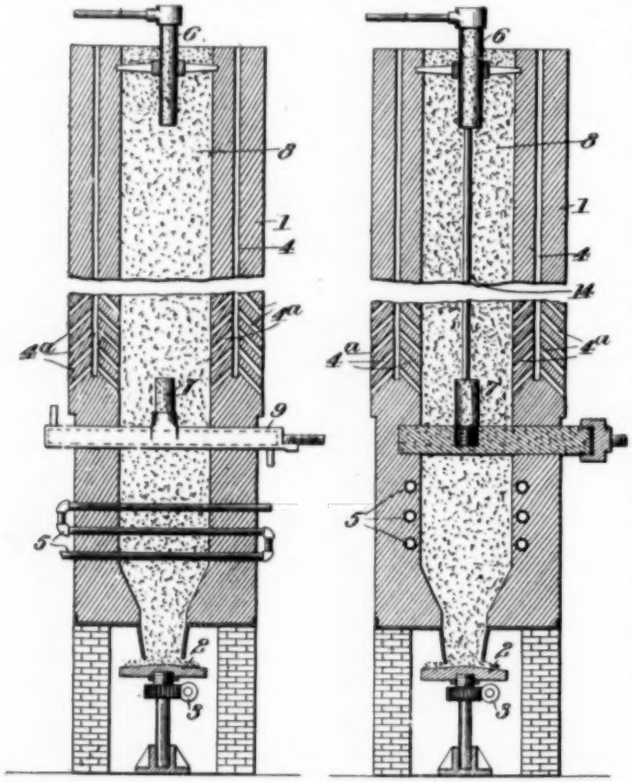


FIG. 14.—CONTINUOUS FURNACE FOR HEAT TREATMENT OF CARBON.

Bulk Graphite.—In the conversion of carbon into graphite in electric furnaces it is necessary to shield the carbon from the direct action of the air, but to permit the free escape of the distillation gases. For this purpose the carbon is surrounded by a granular mass of carborundum, white crust, etc. Under usual operating conditions it is impossible to maintain a sharply defined demarcation plane between the carbon and the surrounding material, especially near the upper and lower surfaces of the charge. It is, therefore, usually necessary, in unloading each furnace charge, to discard a considerable mass of graphite, by reason of the admixed carborundum, etc. Mr. Alfred L. Robinson, of the International Acheson Graphite Company, overcomes this difficulty by the furnace construction shown in Fig. 15. The crushed petroleum coke or gas coke 2,

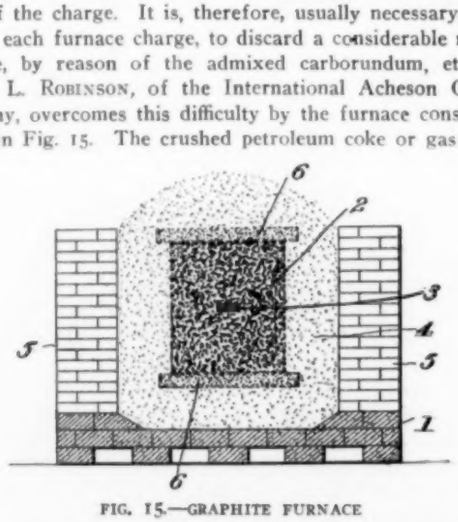


FIG. 15.—GRAPHITE FURNACE

which is to be changed into graphite, is placed between two lower and upper layers 6 of a mixture which during the furnace treatment cakes into a strong solid mass (for instance, a mixture of 25 parts of bituminous coal, 50 ground coke and 25 ground furnace crust, consisting of carborundum, siloxicon, etc.). 3 is a starting core of carbon, used when the initial conductivity of the charge is low; 4 is the embedding material supported by the walls 5 of loosely assembled firebrick. In charging the furnace admixture of the coke with the embedding material is avoided by the use of vertical separating walls of paper. At the close of the operation it is easy to take the furnace to pieces. (1,004,469, Sept. 26, 1911.)

Abrasive and Refractory Material of Alumina and Magnesia.—Research work carried out by the Carborundum Company, of Niagara Falls, has resulted in the production of several interesting new products, for abrasive and refractory purposes, which are disclosed in some patents of Mr. Thomas B. Allen. All of them have alumina as chief constituent. The first product consists essentially of alumina and magnesia, with not over 25 per cent magnesia. Bauxite is used as source of alumina and magnesite as source of magnesia. They are calcined to remove the water from the bauxite and the carbon dioxide and water from the magnesite and the calcined materials are then mixed together with a sufficient amount of carbon to reduce the major portion of the oxide of iron and oxide of silicon. For example, a mixture of 100 parts of calcined bauxite, 15 calcined magnesite, and 6 carbon, treated in an electric furnace, yields a product consisting of 14 parts MgO and 84 Al_2O_3, the remainder consisting of oxide of titanium, iron and silicon. By keeping the temperature of the electric furnace relatively low, the oxide of iron and silicon can be reduced without affecting the oxides of aluminium and magnesium. The resulting ferrosilicon can be separated. The content of magnesia in the finished product affects its toughness. It should not be more than 25 per cent, and by varying it a series of products of varying hardness and toughness can be prepared. (1,001,497 Aug. 22, 1911.)

Abrasive and Refractory Material of Alumina, Magnesia and Boric Oxide.—In a second patent of Mr. Allen boric oxide is added to the alumina-magnesia product. Boric acid is mixed with calcined magnesia, for instance, in the proportion of 35 to 65, and heated in a fuel-fired furnace to about 1300 deg. The product is crushed and 25 parts of it are mixed with 100 parts of calcined bauxite and 8 parts coke and this mixture treated in an electric arc furnace. The product contains 12 per cent magnesia, 6 per cent boric oxide, and 79 per cent alumina, and is a dark colored crystalline material of great hardness and very brittle. Addition of boric oxide increases the brittleness and hardness. (1,001,570, Aug. 22, 1911.)

Abrasive and Refractory Material of Alumina and Beryllia.—A material consisting of alumina and less than 18 per cent of beryllia (that is, less than represented by the formula Be O Al_2O_3) is the subject of a third patent of Mr. Allen. He treats in an electric furnace a mixture of 100 parts by weight of calcined bauxite, 75 parts of beryl, 12 parts of coke (to reduce the oxides of silicon and iron) and 50 parts of oxide of iron (to remove more readily the reduced silicon and titanium). The product contains 7½ per cent of beryllia and 89 of alumina. The content of beryllia gives to the product an improved crystallization, while the toughness decreases with increase of beryllia and the hardness increases with increase of beryllia. (1,001,571, Aug. 22, 1911.)

Purification of Abrasive and Refractory Materials.—In a fourth patent of Mr. Allen a method is described for removing aluminium carbide and ferrosilicon, which are the chief impurities in abrasives produced in an electric furnace. Sodium chloride under the action of heat is used as the purifying agent. The crushed abrasive material is mixed with sodium chloride and heated to a temperature of from 800 to 1200 deg. C. This is followed by washing with water or a dilute solution of a mineral acid, in order to remove any excess of sodium chloride or products of the reaction. Two to five per cent by weight of sodium chloride added to the fused alumina are sufficient for purification. The reactions which eliminate the ferrosilicon and aluminium carbide are given as follows:

$$Fe_4\,Si + 16\,Na\,Cl = 4\,Fe\,Cl_3 + Si\,Cl_4 + 16\,Na$$

and

$$Al_4\,C_3 + 12\,Na\,Cl = 4\,Al\,Cl_3 + 3\,C + 12\,Na$$

These reactions are reversible, going from left to right at high temperatures. If they are carried out at a sufficiently high temperature in an oxidizing atmosphere, then, in spite of their reversibility, these reactions can be carried through to a finish, because the $Fe\,Cl_3$, the $Si\,Cl_4$, etc., are volatilized as soon as formed and the Na and C are oxidized, thus removing all newly formed compounds (those given on the right hand of the equation) from the sphere of action, and permitting the above reactions to go on continuously without stopping at some intermediate equilibrium. (1,001,572, Aug. 22, 1911.)

Purification of Emery Ore.—Emery ore consists principally of alumina, but is always associated with impurities such as iron oxide and usually silica and often small quantities of titanium oxide, etc. To remove these impurities, Mr. FRANK J. TONE, of the Carborundum Company, of Niagara Falls, employs an electric-furnace process, the impurities being reduced by means of carbon. It is important not to reduce any alumina, since in this case an entirely different, easily friable product would be obtained. For this reason only sufficient carbon is used to reduce the impurities. Further, the temperature is maintained above the reduction temperature of silica, but below that of alumina. As this would be impossible with an arc furnace, a resistance furnace is used. The mass of emery ore and carbon is placed between the electrode terminals of a furnace and current is passed through it. As the heating of the mass progresses and it becomes partially fused, its conductivity increases, and when it reaches a well-fused condition the charge becomes a good conductor of electricity. The furnace now acts as a simple resistance furnace. The purified fused alumina collects at the bottom with the impure alumina resting on it on top. The lower layer of pure fused alumina is a poor conductor of electricity, so that the distribution of the current is self-adjusting, the current confining itself practically to the layer of impure alumina on the top. The electrodes are accordingly raised from time to time. (1,002,608, Sept. 5, 1911.)

Boron.—A number of patents of Dr. E. WEINTRAUB, of the General Electric Company, deal with an elaborate method in producing pure boron. The usual method for producing boron includes the reduction of boric anhydride with magnesium, the general process being to heat a mixture of magnesium with an excess of boric anhydride. This reaction is accompanied by the production of magnesium borate and by the formation of other compounds. The resultant product, therefore, contains a relatively high percentage of material other than boron. The usual washing with water and hydrochloric acid decreases the percentage of impurity, but does not yield a product which could be termed, in the chemical sense, "pure boron." This impure product can be greatly improved, however, by "fractional decantation," consisting in grinding the product with water or hydrochloric acid and then decanting off the top of the liquid before complete precipitation has occurred. Boron has a greater specific gravity than the impurities, such as magnesium boride. By repeating this treatment, the proportion of impurities can be reduced several per cent.

The resulting product, in form of a brown powder, is compressed into the form of a stick or rod, say 6 cm long and 5 mm in diameter. The stick is then highly heated in an electric vacuum furnace to dissociate and drive off various of the impurities, like magnesium boride and boron hydride. This treatment may be continued for several hours at a temperature of say 1200 deg. C. without any danger of contaminating the boron by the carbon of the resistance heater. As a result of this treatment, an important physical change takes place, since the boron stick which before the treatment was a non-conductor becomes a relatively good conductor of electricity.

The next step in this process consists in melting down this conductive boron by heating it as anode in a vacuum arc. For this purpose an electric vacuum furnace is used in form of a hermetically closed glass vessel, with some mercury as cathode and the boron stick as anode. Hydrogen gas is introduced into the vessel and a pressure of the hydrogen of from 2½ to 15 cm of mercury is maintained. This hydrogen gas is chemically inert, but serves to concentrate the arc on the lower end of the boron stick.

While chemical text-books describe boron as being a non-conductive material which vaporizes at high temperatures without melting, the boron produced by the above process is a relatively good conductor of electricity, is a dense and solid body and undergoes complete fusion without apparent vaporization. Such pure conductive boron may be used for a variety of purposes and may be shaped into rods or glow bodies for use in incandescent lamps and furnaces. (997,879; 997,880; 997,881; all of July 11, 1911.)

Vacuum Furnace for Treatment of Refractory Materials.—The same principle of melting a refractory material as anode in a vacuum mercury arc may be applied to the treatment of refractory materials other than boron. For instance, for the fusion and purification of tungsten, thorium, osmium, etc., in form of rods pressed from the powdered metal. Several vacuum furnaces for this purpose are described by Dr. Weintraub. The principle is the same in all. (997,882 and 997,883, July 11, 1911.)

Distillation Process for Antimony and Arsenic.—Distillation at reduced pressure is one of the features of a process for treating ores containing antimony and arsenic, devised by Dr. COLIN G. FINK, of the General Electric Company. It may be applied, for instance, to electrolytic slimes containing antimony, and a very pure antimony can be obtained from a slime containing 35 per cent of antimony, the yield being almost 95 per cent. At atmospheric pressure the vaporization point of antimony is between 1500 and 1700 deg. C. At a pressure of 5 to 6 mm mercury it is only 735 deg. C. At a pressure of 40 mm mercury, as used in the author's process, it is between 1200 and 1300 deg. C. For the treatment of stibnite, $Sb_2\,S_3$, using an ore containing 62 per cent of antimony, 11.3 parts of stibnite, 12 parts of limestone, 2½ parts of coke and 6 parts of fluorspar are ground to 20 mesh and charged into an air-tight electric resistance furnace, the charge serving as resistor. The temperature is raised to 600 or 700 deg. C. and maintained there for one and a half hours to let the following reaction proceed:

$$Sb_2\,S_3 + 3\,Ca\,CO_3 + 6\,C = 2\,Sb + 3\,CaS + 9\,CO$$

A vacuum of 40 mm mercury is maintained by constant operation of the air pump. Then the temperature is raised to 1300 deg. C. and maintained there for about one hour, while the antimony is distilled off and condensed in a water-cooled condenser which is in communication with the furnace chamber.

The reduction of the stibnite may also be accomplished by heating the powdered ore with finely divided iron. Two parts by weight of stibnite are mixed with one of iron. The temperature of the charge is raised to 700 deg. C. for 20 minutes, to reduce the stibnite:

$$Sb_2\,S_3 + 3\,Fe = 3\,Fe\,S + 2\,Sb.$$

The temperature is then raised to 1200 or 1300 deg. C. for an hour and the antimony distilled and condensed. Instead of metallic iron, an oxide ore, as hematite or magnetite, may be used, the oxygen of the hematite serving partly to unite with the sulphur of the stibnite to form sulphur dioxide, SO_2 and the remaining sulphur combining with the reduced iron to form sulphide of iron. From two to three parts by weight of powdered carbon to one part of ore are added to the charge, to increase its conductivity and to permit the carbon monoxide formed to keep the charge porous.

Instead of an iron ore, a chrome iron ore, like chromite, may be used, $Fe\,O\,Cr_2\,O_3$. For example, about four parts of finely gound stibnite are mixed with two parts of finely ground chrome iron ore. One part of powdered coke or charcoal is added for the purposes mentioned. The finely ground mixture is heated to 900 deg. for one or two hours, to reduce the antimony and chromium:

$$3\ Fe\ Cr_2\ O_4 + Sb_2\ S_3 + 12\ C = 3\ Fe\ S + 12\ CO + 2\ Sb + 6\ Cr$$
$$2\ Fe\ Cr_2\ O_4 + Sb_2\ S_3 + 6\ C = 2\ Fe\ S + SO_3 + 6\ CO + 2\ Sb + 4\ Cr.$$

The temperature is then raised to 1500 deg. C. for one or two hours to distil and condense the antimony. (996,474, June 27, 1911.)

Silica Glass.—Mr. H. A. Kent, of the Silica Syndicate, of London, England, patents a process of making silica glass (fused silica), the first claim reading as follows: "The manufacture of silica glass by causing grains or particles of silica to fall or be projected separately through a heated zone and onto a movable heated body in a chamber, said particles being subjected to a temperature sufficient to sinter them by this single operation into a homogeneous mass of silica glass." One method of making a silica rod or tube is shown in Fig. 16.

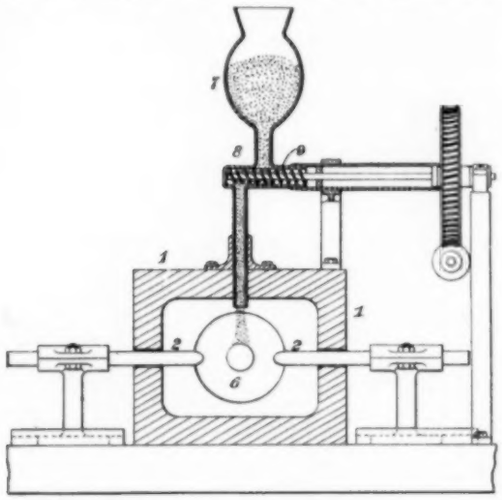


FIG. 16.—MANUFACTURE OF SILICA GLASS.

The electric furnace 1 is provided with adjustable electrodes 2. The ground silica grains are carried from the container 7 through the tube 8 with the revolving worm 9 and drop down onto the heated cylinder 6 of silica-glass upon which they form a homogeneous mass of further silica glass. (1,003,271, Sept. 12, 1911.)

Burning Cement with Electric Heat.—Methods have been proposed for burning cement by the direct heat of the ordinary short arc, but for practical reasons the scheme has not been adopted. Mr. Carleton Ellis, of Montclair, N. J., has recently patented a process for the utilization of a long arc in cement burning. An alternating current of from 4000 to 6000 volts is used, and this for commercial reasons may be a three-phase system. Three or more arcs may be arranged to work on the three phases of such a system, with the kiln connecting with the neutral junction. With a kiln about 20 ft. long and about 20 in. in diameter, a current of from 200 to 300 amp and at 4000 to 6000 volts may be used. The length of the kiln determines the voltage necessary to strike the arc, while the amperage is adjusted in accordance with the heating effect required.

In order to prevent the arc "blowing out," the inventor injects a spiral current of air moving tangentially around the arc, as by this means he is enabled to project arcs of great length without loss of stability. In order to protect the lining of the kiln as much as possible, the kiln may be rotated at high speed so as to carry the load well up on the side of the kiln, and by the application of the electrode magnet the arc may be deflected at one or more points into close contact with the material to be clinkered, and away from the opposite and unprotected side of the kiln. (999,494, Aug. 1, 1911.)

Gas Reactions.

Nitric Acid from Air.—The Pauling process for the fixation of atmospheric nitrogen by the electric arc has come into considerable prominence by the two commercial plants in Austria and France employing it on a large scale (this journal, Vol VII, p. 430; Vol. VIII, p. 102, 196) and also by the decision to use it in a plant in course of erection in the South. In the existing Pauling plants the arrangement is very similar to that of the horn lightning arrester, as described in the articles referred to. It appears, however, from two recent patents of Mr. Harry Pauling that he is also paying attention to quite different designs, using a long vertical arc. But the arrangement is very different, for instance, from the long arc of the Badische Company's furnace. While in the latter the air passes in spiral form along the whole vertical arc from the bottom to the top, Pauling provides gas inlets and outlets alternately all along the arc. The furnace has a central vertical open shaft within which the arc plays. The walls contain gas outlets alternately from the bottom to the top. When the air enters through one of the inlets it travels only a comparatively short distance along the arc, during which time the nitrogen combines with the oxygen, and the gas mixture formed is then withdrawn. The principal advantage claimed for the arrangement is that it is possible to reduce the velocity of the air current to a value which obviates all risk of blowing out the arc. (999,586, Aug. 1, 1911.)

A second patent of Mr. Pauling refers to the difficulty experienced with a long arc in a comparatively narrow shaft on account of the fact that the walls of the shaft being heated to a high temperature may become conductors and carry some of the current. This is prevented by subdividing the surface of the furnace walls facing the flame into a number of areas or elements, each of which is separated from the surrounding elements by gas passages which penetrate into the wall to such a depth as to obviate any risk of the wall acting as a conductor when it becomes hot. (999,587, Aug. 1, 1911.)

The long vertical arc of the Schoenherr furnace of the Badische Company was originally caused to play between a solid iron core as lower electrodes (where the air enters) and a hollow cooled upper electrode (where the gas mixture leaves). In their latest construction Dr. Otto Schoenherr and Mr. J. Hessberger employ a hollow cooled electrode also at the lower entrance end of the furnace, arranged in such a way that the starting point of the arc at this lower electrode moves about on its surface, while the arc as whole remains stable and stationary. (1,003,299, Sept. 12, 1911.)

In a patent of Mr. Johan Jetmund Thoresen, of Kristiania, and Mr. Filip Tharaldsen, of Meraker, Norway, the method is patented for treating air by means of an arc made to rotate by a rotary magnetic field. The process is described by reference to the induction motor in which the primary pole pieces produce a rotary magnetic field and this rotating field causes the secondary (the rotor or the armature) to rotate. "The electro-dynamic action between the electrically rotatory primary field and the current induced in the induction coils produces a torque, which causes the secondary core to rotate along with the said field. By preventing the secondary core as well as the coils from rotating and by making a rupture in the said coils and placing the ends of the rupture (electrodes) in the field perpendicular to the direction of the latter, an electric arc or series of electric discharges will be produced in the space between the cores, and the current will pass through said arcs perpendicularly to the field. The electro-dynamic force produced by the combined action of the current in the arcs and rotary field will cause the arcs to be deviated to one side and in the direction in which the field rotates. As the direction of the lines of force of the rotary field is determined by the shape of the field cores and the path of the arc being always perpendicular to the field, it is evident that by giving the field cores a cylindrical or other shape and moving the field accordingly the arcs will assume a cylindrical or other shape. By making the cores cylindrical and causing the field to rotate, the elec-

trodes being placed at suitable intervals, we obtain a continuous row of arcs, each forming a part of a cylindrical surface, and all the said arcs together constitute a complete cylindrical surface. The arc may be supplied with current from a separate single, two or three-phase transformer or from a source of direct current instead of supplying the same with current from the secondary coils. It may be necessary to have in series with the arcs a certain amount of self-induction in order to impede the flow of current. We can also connect the electrodes in series with the coils that produce the magnetic field." (996,951, July 4, 1911.)

Mr. CARLETON ELLIS, of Montclair, N. J., uses a long arc and causes it to revolve about its normal axis in the path of the current of air and at a slightly lower speed than the current of air. The extinguishing effect of the air blast is minimized by the almost equal speeds of arc and air, while "the aureola contact is greatly reduced over methods which involve the travel of the air longitudinally with the arc. The process therefore yields a large output with simplicity of furnace construction, small number of units and therefore reduced labor cost and maximum electrical efficiency." (1,002,249, Sept. 5, 1911.)

Ozone Generators.—The details of construction of an ozonizer, invented by Mr. LOUIS GOLDBURG, of Indianapolis, Ind., are well described in his third claim: "An ozonator, comprising a casing having a contracted inlet opening at one end, a block fitting within said casing adjacent to said inlet and having therethrough a plurality of distributed air passages communicating with said inlet, a cylindrical shoulder formed on said block, a tubular dielectric fitting said shoulder and adapted to be supported thereby in said casing, an outer electrode disposed around said dielectric tube, an inner electrode having screw-threaded connection with said block and located concentrically within said tube, said electrodes being formed to provide opposed projections distributed throughout the contiguous areas of said electrodes, the projections on one of said electrodes being uniformly spaced away from those on the other, the end of said casing which is opposite said contracted air inlet being arranged to permit said block, dielectric tube and electrodes to be readily withdrawn therethrough, and means for connecting said electrodes with a suitable source of electromotive force." (995,958, June 20, 1911.)

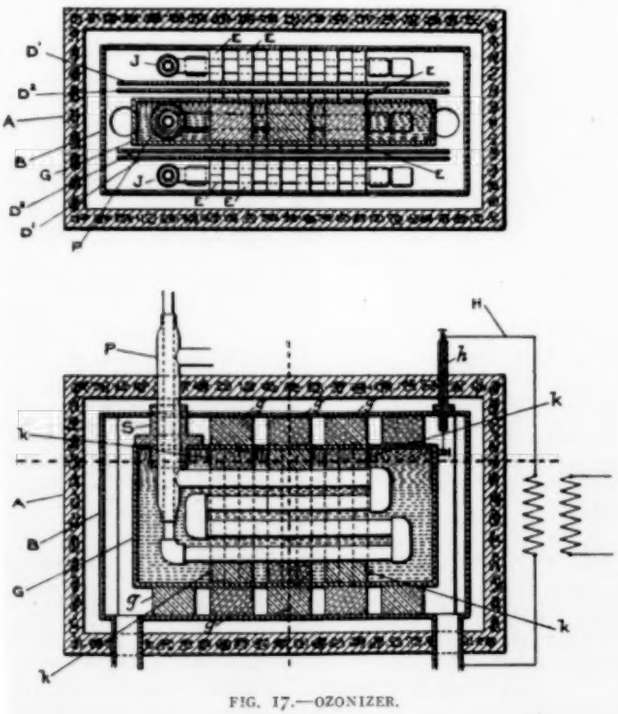


FIG. 17.—OZONIZER.

In an ozonizer of Mr. JAN STEYNIS, the chief feature is the elimination of some of the heavy insulation usually required. This object is obtained by grounding one side of the high-tension circuit and also grounding the cooling apparatus. Fig. 17 shows a plan and a central longitudinal vertical section of the ozonizer. *A* is the external casing, *B* is a box of thin metal connected to the grounded side of the circuit. On each of the longer sides of box *B* and on the inside thereof are a series of vertical metallic bars *E*, *E*, etc., which constitute the discharge points of the electrode. The bars *E* are perforated at intervals in a direction parallel to the longer side of the box, and through these perforations extend a series of metallic pipes forming a cooling coil *J* for receiving a gasified liquid, or in some cases cold water or brine. Since the said box *B* is grounded, the pipe coil need not be insulated as heretofore. Against the faces of the discharge bars *E* is placed a thin sheet of glass or micanite D^1 which is separated a slight distance from a parallel sheet of like material resting against the faces of similar discharge bars on the outside of the central electrode. Between these sheets passes the air to be ozonized.

The central electrode consists of an internal metallic box *G* with the vertical discharge bars *E*, *E*, etc., on its outer sides. The box is insulated from the surrounding box by blocks, *g*, *g*, etc., and is connected to one side of the circuit by a wire *H*, passing through and insulating tube *h* in the top side of the box *B*. Within the central metallic box *G* is a cooling coil *P*, like the coils *J*. It is, however, insulated by oil or other liquid insulator from the box *G* in which it is contained, being supported in the box by insulating blocks *k*, *k*, etc., and where it passes through the top side of the box the outgoing end is enlarged to receive within it the incoming end, and thereby call for only one insulating bushing *S*. The coil *P* will be connected to the coils *J* and to ground, so that the only insulation required in the apparatus is that for the central electrode. (997,339, July 11, 1911.)

Electrolytic Furnaces.

Partition for Electrolytic Furnaces.—In electrolytic furnaces partitions are sometimes required, for instance, for separating the anode gases from the metal deposited on the cathode. It has been proposed to form such non-conductive partitions by chilling a layer of the molten electrolyte itself against a water-cooled metallic curtain. However, it is not possible to maintain such layer of chilled electrolyte of sufficient thickness and at a sufficiently low temperature to avoid its conducting a portion of the current from the molten electrolyte to the curtain. Messrs. GEORGE O. SEWARD, of East Orange, N. J., and FRANZ VON KÜGELGEN, of Holcombs Rock, Va., in a patent assigned to the Virginia Laboratory Company, overcome this difficulty by using two curtains, side by side but insulated from each other,

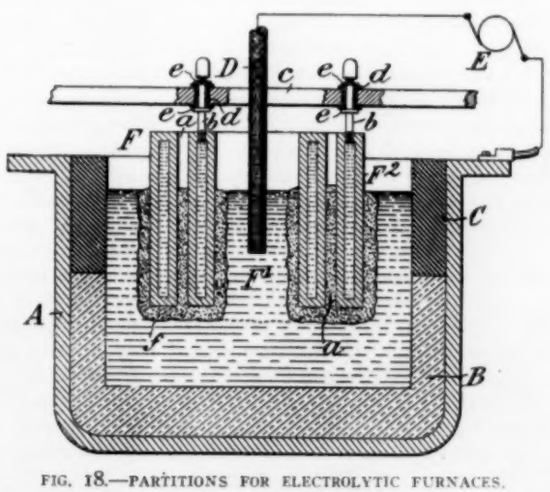


FIG. 18.—PARTITIONS FOR ELECTROLYTIC FURNACES.

instead of a single one. In Fig. 18 *A* is the circular electrolytic vessel or furnace with an insulating lining *B* in its lower part and a conductive carbon lining *C* at its upper part, which serves as anode. In the center of the furnace is the cathode *D*. Between anode and cathode is provided the non-conductive partition *F* which consists of two tubular concentrical cylindrical water-cooled curtains F^1 and F^2, between which is an annular

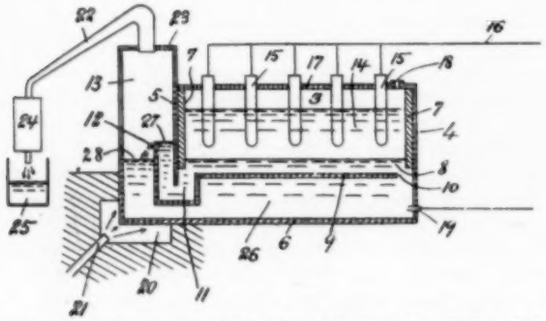


FIG. 19.—PRODUCTION OF METALLIC SODIUM FROM FUSED SODIUM CHLORIDE.

space *a*, which is kept filled with congealed salt or packed with insulating material, such as fire-clay or asbestos. (13,277, July 25, 1911.)

Metallic Sodium by Electrolysis of Fused Sodium Chloride.—In the electrolysis of fused sodium chloride with a cathode of molten lead the sodium set free at the cathode alloys with the lead and it is necessary to remove the sodium-lead alloy from contact with the molten electrolyte for two purposes: First to provide a fresh molten lead cathode surface ready to alloy with sodium, and second to recover the sodium from the sodium-lead alloy in the form in which it may be desired, for instance, as metallic sodium. For this removal of the lead-sodium alloy from the electrolytic cell, Mr. Robert J. McNitt, of Niagara Falls, N. Y., in a patent assigned to the Roessler & Hasslacher Chemical Company, makes use of differences of specific gravity to provide the circulation of the molten electrode material. Pure molten lead has a higher specific gravity than a molten alloy of lead and sodium. If, therefore, in Fig. 19 the channel 26 is first filled with molten pure lead and the compartment 8 (just above the partition 9) and the seal 11 with a molten alloy of lead and sodium on which rests the molten electrolyte 14 of sodium chloride, there will be a difference between the hydrostatic levels at the points 27 and 28. When a direct current is then passed from the anodes 15 through the sodium chloride 14 to the lead cathode 8 (electrical connection being made at 19 to the external circuit), the sodium chloride is decomposed, the sodium alloying with the molten lead 8. More sodium chloride is now fed into the cell, etc. Meanwhile, of course, a portion of the cathode alloy is forced down through seal 11 under wall 5, over wall 12 into the outside furnace chamber 13. In this chamber 13 the temperature is maintained high enough so that some or all of the sodium in the lead-sodium alloy is driven off by distillation and is condensed in 24. There remains back in 26 a weak alloy which returns through opening 10 into the electrolytic chamber 3. The difference of specific gravity is, therefore, utilized to maintain a difference of levels at 27 and 28 and therefore provide an overflow from 27 to 28. (995,476, June 20, 1911.)

Electrolytic Processes.

Mercury Cathode Cell.—A patent of Mr. Meyer Wildermann, of Ealings, London, England, refers to improvements in his mercury cathode cell, in which the mercury is contained in superimposed troughs (see our vol. I, page 589). The details of improvement are indicated in the first claims: "In electrolytic cells for the electrolysis of alkaline salts provided with a mercury cathode contained in superimposed troughs, with division ribs dipping into the mercury, with stirrers for mixing the mercury and amalgam on the anode side, with carbons floating on the mercury on the cathode side, the provisions of means for preventing the carbons from rising over a certain height in the troughs. (998,063, July 18, 1911.)

White Lead.—A process differing in various respects from his former white-lead process is the subject of a patent of Dr. Carl Luckow, of Cologne, Germany. While the specification

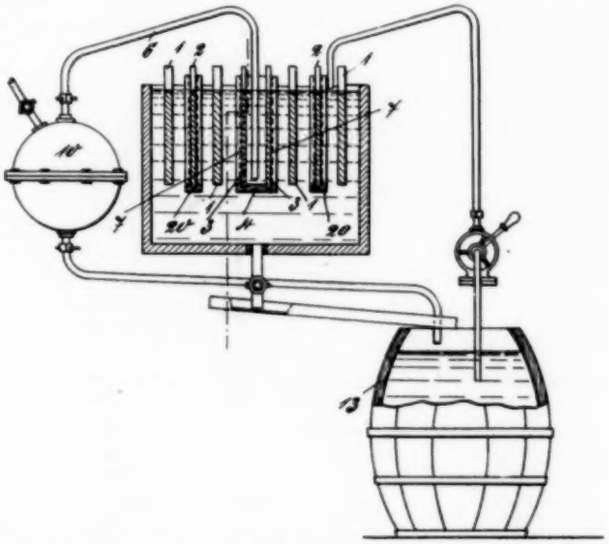


FIG. 20.—ELECTROLYTIC PRODUCTION OF WHITE LEAD.

relates in general to the electrolytic manufacture of difficultly soluble salts of heavy metals and gives data on the production of chrome yellow, Prussian blue and Paris blue and white lead, only the white-lead process will here be described. The anodes 1 of Fig. 20 consist of soft lead. The cathodes 2 are skeletons of soft lead or hard lead with active fillings of spongy lead oxide. To prevent enrichment of the electrolyte with caustic alkalies, the dilute solution caustic, as it is being formed during electrolysis, is constantly withdrawn by means of what is called "filtering cathodes" 3, 4. All the cathodes 2, including the filtering cathodes 3, 4, are inclosed with wooden frames 20 and filtering cloths 7. The electrolyte is a diluted solution containing from 1 to 2 per cent of a mixture of from 90 to 95 parts by weight of sodium chlorate and 10 to 5 parts of sodium carbonate calculated free from water. During electrolysis the carbonate and water are always added as they are consumed, while the soda-lye is always extracted, as it is

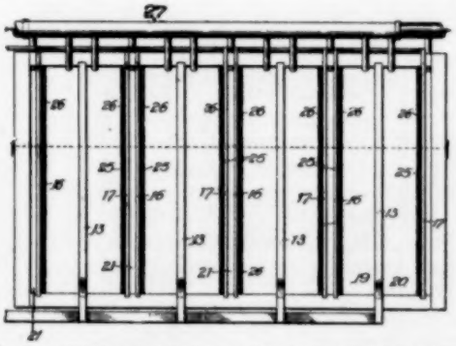


FIG. 21.—ELECTROLYTIC TREATMENT OF WASTE LIQUORS.

produced, from the filtering cathodes 3, 4, by means of a syphon 6 and passed through a receptacle 10 for carbonic acid into the tank 13, from whence it is fed back into the cell. (1,006,389, Oct. 17.)

Electrolytic Recovery of Waste Liquor.—In a patent assigned to the National Tube Company, Mr. Fred. F. Farnham, of McKeesport, Pa., recovers iron and sulphuric acid

from solutions of ferrous sulphate resulting from cleaning steel by means of sulphuric acid. He uses a diaphragm cell, in which diaphragms of asbestos 25 with an outer layer of burlap 26 (see Fig. 21) are fastened to the perforated lead anodes 16 and 17. The iron sheets 13 are the cathodes. Between each cathode 13 and the adjacent anodes 16 and 17 are formed the cathode chambers 19 and 20, while between adjacent anodes 16 and 17 the anode chambers 21 are formed. During electrolysis sulphuric acid accumulates at the anodes and iron is deposited on the cathode. The ferrous sulphate solution is fed from the inlet pipe 27 into the cathode compartments. It gradually seeps or filters through the diaphragms and the associated anodes into the anode compartments. While the solution is in contact with the anodes, sulphur dioxide is formed with the consequent formation of sulphurous acid, which is immediately oxidized to sulphuric acid, the reactions being $SO_2 + H_2O = H_2SO_3$ and $2\,H_2SO_3 + O_2 = 2\,H_2SO_4$. This sulphuric acid is withdrawn from the anode chamber through the outlet pipe. (1,006,836, Oct. 24, 1911.)

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

Melting Zinc-Box Precipitate.—Some interesting points bearing on the production of high-grade bullion from zinc-box precipitate is given in the August, 1911, issue of the *Monthly Journal* of the Chamber of Mines of Western Australia, by Messrs. J. B. AARONS and HERBERT BLACK. The authors were confronted with conditions which gave them low-grade bullion, and experimented with a view to raising the grade and thereby reducing refining and banking charges.

Their present procedure is as follows: All the precipitate and short zinc are sent to the acid vat, and only the well-washed long zinc is returned to the boxes. No scrubbing or screening of short zinc is attempted. After acid treatment the precipitate is filter-pressed, washed three or four times with hot water, dried with air and dropped into cast-iron trays, weighed and placed in a cast-iron muffle, where it is subjected to a roast at dull red heat, or merely dried. The dried or roasted precipitate is not screened, but is directly fluxed with borax and sand, and in some cases with nitre. A typical charge is 100 parts precipitate, 50 parts borax, 20 parts sand, and when nitre is used, 7 to 10 parts nitre. The charge is smelted in clay liners in plumbago pots and poured into conical molds. The bullion thus obtained is subsequently melted in plumbago pots and cast in bars of 500 oz.

The authors believe that it is easier to produce high-grade bullion direct from the precipitate than to produce low-grade bullion direct from the precipitate and afterward refine it; moreover, there is less chance of loss when the refining is carried out through the medium of the flux than in the refining of the metal. They further believe that if the acid treatment and subsequent washing are carried out thoroughly, roasting to dull red heat is unnecessary, and that drying at 100 degrees Centigrade is sufficient. The production of high-grade bullion is mainly a question of thorough acid treatment, careful washing, and the selection of the proper fluxes in the fusion. The use of fire-clay liners in the plumbago pots is a strong factor in the production of high-grade bullion, whether nitre is used or not, as it guards against the reduction of zinc from the zinc sulphate or silicate present.

Tables of figures are given to show that the use of nitre is not attended with the baneful effects usually attributed to that flux. Equal quantities of precipitate, one fluxed with nitre and the other without, gave results favorable to the use of nitre. Further evidence is offered in tables showing a comparison between the gold called for by assays of the precipitate and the gold recovered in bullion, showing practically no loss of gold.

A comparison between smelting in plumbago pots and clay liners in plumbago pots gives results very favorable to the latter, showing a higher grade and fineness of bullion. The use of nitre, however, is found to be unnecessary when clay liners are used.

As regards roasting to a red heat versus drying of the precipitate, the authors believe that the latter is all that is necessary unless the acid treatment is poorly done, or in the case of coatings of base salts on the zinc. In the latter cases roasting is necessary or else treatment with nitre in clay pots. With regard to the elimination of the base metals, zinc, copper and lead, thorough acid treatment and washing, with reduction in clay pots, should eliminate the zinc; nitre flux and the use of clay liners should remove the copper, and similar treatment should take care of the lead, although manganese dioxide may be added to facilitate operations.

The following deductions are made as a result of the authors' work: No screening of short zinc is necessary, the cost of extra acid being more than compensated for by the reduction of refining and banking charges. Acid treatment should be as complete as possible. Washing in the press is by no means sufficient and leaves much to be desired. Roasting at red heat is unnecessary except as noted heretofore; drying is all that is necessary. Clay liners to plumbago pots are essential to the production of high-grade bullion unless all the zinc salts can be washed out of the precipitate. The use of thees liners gives slags of lower value; more quiet fusion; the absence of obnoxious fumes and lower bullion realization costs.

Losses in Cyanidation.—Under this caption Mr. H. FORBES JULIAN discusses the relative merits of filtering devices, and some errors in their operation, in the October, 1911, *Mining Magazine*. Leaving out of consideration the losses which occur before filtering commences, the author assumes the preparation of a slime pulp of a consistency ranging from 1½ tons to 3 tons of liquid for one ton of solid, although this may be thickened to a consistency of 1 : 1.

Cake Formation and Thickness.—When filtering slime containing much amorphous matter it is well to apply a low suction, about one-quarter of the maximum, until the cake is about ⅛ in. thick, and then apply the maximum. Operators as a rule pay too little attention to this variation in vacuum, but keep it constant throughout cake formation. The thickness to which a cake is allowed to build up is a matter that must be determined by experiment, for it may be only ½ in. in the case of a gelatinous slime, or 2 in. when crystalline material is filtered. The rate at which the filtrate is drawn may be utilized to determine the best thickness.

The porosity of a cake is affected by the pressure of formation. Thus a cake deposited by vacuum contains 40 to 65 lb. dry ore per cu. ft., according to the specific gravity and fineness of the material, while the same material filter-pressed in a cake contains 70 to 110 lb. dry ore per cu. ft. This explains why high-pressure filters give thin, sloppy cakes from which the valuable solution has not been fully pressed, when filtering amorphous material, whereas the vacuum filter working on the same material forms thick, porous dry cakes.

Characteristics of Filter Cakes.—Vacuum filters will produce a thicker cake from so-called natural slime, and one that is more uniform in its porosity; but on crystalline material a pressure filter will produce a heavier cake in given time, but not heavier in proportion to the power expended, and not uniform. The cake formed on a vacuum filter is more uniform, being made up of a number of layers, whereas in the pressure filter, segregation is unavoidable. This makes washing more uniform in the vacuum filter. Moreover, in washing vacuum filter cake a turbid wash solution can be used which will soon adjust the porosity of the cake if there is any tendency to uneven flow.

Care of the Cake.—During the time the cake is exposed to air it must be handled with extreme care. If too high a vacuum is held it has a tendency to disturb the grain of the cake and disturb its even porosity. If suction is cut off entirely there is danger of portions of the cake falling off the filter. Vibrations also must be reduced to a minimum, as these disturb the integrity of the cake.

Washing the Cake.—When the cake is ready for washing

it contains 30 per cent to 35 per cent of its weight of valuable liquor, together with undissolved metallic gold and silver particles and compounds. The liquor may be washed out by water, but if the latter values are to be recovered they have to be removed by a solution wash. The solid gold and silver remaining in the pulp may be still in its natural state, or it may have been reprecipitated during the agitation process. In any event these solid particles dissolve slowly, the latter more so than the former, and therefore the rate of solution washing must be slow enough to give the necessary time for contact between the solution and the particles.

It is well known that in the dissolving and agitating process, a point is reached beyond which the cyanide solution will not dissolve any more gold. This is due to reaching a point of equilibrium due to the presence of various salts in solution. If it were possible to bring fresh, potent solution in contact with the pulp at that time, further solution could take place. This is exactly what is possible in washing a filter cake with barren solution, but sufficient attention is not paid to it.

Reprecipitation and Resolution.—Reprecipitated metal does not dissolve with the same readiness that is noted in the natural state. Reprecipitation is due to local action of physical and chemical forces. Graphite, tellurides, sulphides, arsenides, charcoal, grease, oils, and even silica or alumina in a finely divided state, precipitate gold and silver from working cyanide solutions, and this precipitate shows little tendency to again dissolve in the same solution. In fact, it is difficultly soluble in fresh solutions. Hence in washing a cake with solution for the purpose of removing as much as possible of the undissolved or reprecipitated gold time should be allowed for contact. A cake ¼ in. thick requires about as much time to be solution-washed as does one 2 in. thick.

Preparation and Application of the Wash Solution.—If it is not already aerated it should be treated to give it maximum oxidizing power, and if not already turbid it should have some slime added. It should be made stronger in cyanide than that originally used in the treatment tanks, and some free alkali should be added. In applying this solution it is necessary only to give it such a rate of flow as would prevent backward diffusion, but in practice this time cannot be allowed. Nevertheless, the time allowed should be as great as possible, and not as absurdly low as in some instances.

Water Wash.—The object of the water wash is to remove cyanide solution remaining in the pulp from the solution wash. This would appear to be a simple matter, but in practice it is found that a great deal more water must be drawn through the cake than that indicated by a simple calculation. A low vacuum throughout the whole wash is desirable for a maximum strength of filtrate, but a high vacuum at the start would save time if maintained no longer than is necessary to just withdraw the stronger portion of the filtrate. But even then some cyanide will be held by absorption, requiring time of contact to diffuse into the water wash. The more porous the cake the lower should be the vacuum, and the finer and more amorphous the material the longer should be the time of contact.

Washing by Filtration and Decantation.—The simplest of all methods of washing is by decantation, but in practice there are serious objections to it. The unavoidable loss of dissolved metal remaining in the pulp is the greatest. In filtering, the amount of dissolved metal left in the cake has no relation to the original richness of the solution. In decanting, the metal contents are removed by two to four washes or dilutions, while in filtering this is done by an infinite number of washes. If the total quantity of liquor used in the decantation process be made the same as in the filter process, say 3 : 1, the theoretical separation of dissolved metal by three decantations would be only 78.4 per cent. In practice 75 per cent would be considered fair work. In the filtration process with this same proportion of liquor there is no difficulty in removing 99 per cent of dissolved metal. But in order to get 99 per cent of dissolved metal by the decantation process it would require a total of 12 tons of wash liquor to 1 ton of solid. Wash waters generally contain small quantities of dissolved metal, and the solution in the residue pulp generally contains this metal in proportion to the amount of moisture retained, which for filtration is about 1 of solution to 2 of solid, and for decantation 1 : 1. Hence this loss in decantation is twice what it is in filtration. Again the wide difference in the quantity of wash water used in the two processes affects the cost per ton of ore. In filtration the amount of wash liquor used seldom exceeds twice the weight of the solid, while in decantation it is common to use five of wash liquor to one of solid. Finally reprecipitation and absorption are greater in decantation than in filtration, as the very dilution with water in the decantation process is sufficient to cause silica and alumina to precipitate and carry down gold and silver and cyanide.

The Brakpan Mill, Transvaal.—One of the newer cyanide mills on the Rand is that known as the Brakpan, in which are installed such modern features as zinc dust precipitation, agitation of slime in Pachuca tanks, and filtration in Butters vacuum filters. A complete description is given in the *South African Mining Journal* for Aug. 12, 1911.

The ore is first separated into coarse and fine portions, sorted, and the coarse portion crushed. The battery bins have a capacity of 6,000 tons of fine ore. Stamps of 2,004 lb. weight are used; at present there are 160 installed and provision is made for 40 more. The stamp heads are 42 in. long, 9½ in. in diameter and weigh 788 lb. Stems weigh 696 lb., and are 14 ft. 6 in. long and 4½ in. diameter. Tappets weigh 232 lb., and a shoe weighs 288 lb., and is 14 in. long and 9¼ in. diameter. Ten stamps are driven by one motor.

The stamped ore is distributed to the tube mill cones, which are of the Caldecott diaphragm type. There are six tubes, 5 ft. 6 in. by 22 ft., and provision is made for the installation of four more. The feed contains 35 per cent moisture. Following each tube is a set of five stationary amalgamation plates, 12 by 5 ft., with a grade of 18 per cent. The amalgamated pulp is classified in cones, the overflow going to the cyanide plant, while the underflow is returned to two of the tubes for further grinding.

The cyanide work is divided into separate treatment for sand by percolation, and slime by agitation. The pulp delivered to the cyanide plant is, therefore, classified into sand and slime by means of cones. The sand is delivered to the collection tanks with 30 per cent moisture, but additional water is added, so that clean sand is delivered to the 60-ft. collectors. The overflow of the collectors is again cone-classified to remove the last traces of sand from the slime. From the collectors the sand is conveyed by belts to a Blaisdell distributor which serves the treatment tanks. After treatment the sand is removed by a Bleichert system of aerial haulage running from beneath the tanks to a cantilever tower where it is dumped. This system was installed with a guarantee to dump tailings at a cost not to exceed three cents per ton.

The slime overflow of the classifying cones passes to slime collectors where it is allowed to settle. It is treated in six Pachuca tanks, 15 ft. by 45 ft., where four hours agitation is given. Stock tanks are placed between the Pachucas and the Butters Vacuum filter, the latter consisting of two boxes and a total of 336 leaves. The filtered solution is clarified in two sand filters, used alternatively, and passes to gold solution storage tanks. Precipitation is made by zinc dust in Merrill presses. The zinc emulsion is made in emulsifying cones in which the zinc dust is agitated with weak solution before being added to the gold solution. The solution from sand and slime treatment together with the zinc emulsion, is pumped to the Merrill presses by three plunger pumps driven by 10-hp motors.

Electric power is used throughout the plant, the aggregate power of motors installed being 10,000 hp for mine and mill. In the mill the distribution of motors is as follows: Stamp battery and tube mills, 1950 hp; reductions works pumping plant, 1000 hp. The power supply is delivered through underground cables, paper insulated and lead covered. These cables are laid

on a ring system, so that supply can be maintained to any point through an alternative part of the system in the event of a break in any particular cable.

Zinc.

Electric Zinc Smelting in Sweden and Norway.—For several years extensive experiments have been carried out on the use of the electric furnace in zinc metallurgy in Sweden and Norway, as has been described repeatedly in this journal, but details have been maintained secret with great care. However, now information regarding the process has become available for the first time, owing to the controllers of the works seeking English and German capital for the purpose of continuing the trials and enlarging the plant. According to the October issue of the *Mining Magazine*, of London, an English company has been formed for the purpose of acquiring two electric smelting works, one at Sarpsborg, Norway, and one at Trollhättan, Sweden, and also the rights in connection with the construction of another at Tysse, near Odde, in Norway. The process and furnace used at these works are intended for the treatment of complex sulphide ores and other material, such as Broken Hill slime and the intimately mixed sulphides of lead, zinc, copper and iron that occur in large quantities in various parts of the world, at present of little or no commercial value. Reports have been made by J. C. Moulden for the Sulphide Corporation and by F. W. Harbord, the latter of whom made thorough tests of the plants at Sarpsborg and Trollhättan. The information here given is called from the reports of these experts:

"The electric furnace employed has carbon electrodes and the heating and melting of the charge is effected by resistance with a current of low voltage. One electrode is in the bed and the upper one is removable through the roof. The cross-section of the upper electrode measures approximately 2¼ sq. ft. In the first furnace the charge was introduced through the roof, but more recently the door is placed in the side and the operation is thus made more continuous. The capacity is 3 tons, and 2.8 tons of roasted ore can be treated per 24 hours. The horse-power per furnace is 350, and at Trollhättan the cost of power supplied by the hydroelectric company is 30s. 3d. ($7.56) per horse-power year. At Mr. Harbord's tests on roasted Broken Hill slime the energy used per ton of ore was 2078 kw-hours. There are 11 furnaces in Trollhättan. The roasted ore, coke or anthracite dust and flux are charged into the furnace, and most of the zinc and some of the lead is volatilized. The greater proportion of this is condensed as a crude spelter with high lead content, but much of it is precipitated as metallic fume and oxide, and has to be returned to the furnace in subsequent charges. The lead that is not volatilized is reduced as liquid metal, and it contains a considerable proportion of the silver. Matte is formed of the iron, copper and sulphur, if present in the ore, and some of the lead, zinc and silver passes into it. Some lead also escapes with the slag. The crude zinc is refined in other electric furnaces and the final zinc product is exceptionally pure, averaging 99.9 per cent and commanding a premium price in the market."

"At the time Mr. Harbord made his examination the furnaces were built of porous material, apparently similar to that used in old-fashioned zinc distilling retorts, and some of the lead and silver percolated through and had to be caught underneath; this he calls 'leakage' lead. He found also that the lead that exuded in this way during his experiments did not necessarily come from the charges under his observation and was partly produced during the treatment of other ore at an earlier period. Thus the silver contents of the lead did not at all coincide with that of the ore treated by him. The general results of his trials were by no means ideal, and he makes many suggestions for improvements; but his conclusions are that the process may easily be developed into one of prime importance, especially at the new plant proposed at Tysse, where the estimated cost of a horse-power-year is only 14s. ($3.50).

"During his experiments at Trollhätten seven furnaces treated during 27½ days, 518 tons of roasted Broken Hill slime, 19 tons of calamine, 22½ tons of zinc-lead powder produced previously, and produced 160.8 tons of crude zinc which by refining yielded 112.4 tons of pure zinc and 24.7 tons of lead, together with 36 tons of powder, in addition to which 41 tons of lead were tapped containing 141 oz. silver per ton and 'leak' lead amounting to 17 tons was recovered assaying 27 oz. silver per ton; also 9 tons of skimmings containing zinc, lead and silver. The recovery from these various sources was 64 per cent of the zinc, 74 per cent of the lead and 40 per cent of the silver, not reckoning the metals recovered in the powder form. Mr. Harbord goes into details as to many points where increased extraction and lower costs may be obtained."

"Mr. Moulden is of opinion that, by means of improvements already indicated, the extraction should eventually be 75 per cent of the zinc, 80 per cent of the lead and 80 per cent of the silver, and on this basis he figures the profit obtainable on slime bought from the Sulphide Corporation and shipped to Sweden. The slime is taken to average 32 per cent zinc, 23 per cent lead and 25 oz. silver, and on the current basis of payment would cost £2 16s. 8d. per ton at Broken Hill. The freight to Trollhättan would be £2 5s. 3d., making the total cost per ton at the smelter £5 8s. 5d. The value of the recovered metals on the percentage figured by Mr. Moulden would be £9 9s. 11d. The cost of treatment he estimates at £3 0s. 7d., so that the margin of profit would be £1 9s. ($7.25) per ton of slime. It is unnecessary to say that the process is in its infancy and that already many improvements are obvious; also that everything depends on the cost of electric heat. Enough is shown, however, in these reports to make it clear that the process and method of metallurgy deserve close attention."

The subject is discussed by the *Mining Magazine* in an editorial which gives figures on the capitalization of the companies and contains the remark that "the present cost of electrodes is staggering, amounting to £204 ($1020) during the treatment of the 550 tons of material" in Mr. Harbord's tests. Attention is drawn to the fact that a great many other electrometallurgists are experimenting on the same lines and that "theoretically their furnaces both in principle and in method of construction are practically identical with those at Trollhättan, so the relative skill of manipulation, attention to detail, and cost of power will decide the actual success of any of these rival processes." It is further pointed out that "the practice in zinc metallurgy in retorts had been so much improved during the last year or two that the lead content is no longer a drawback; in fact, the buyers desire to have at least 5 per cent lead, for this is found useful in collecting the silver, and the leady residue left in the retorts is sent to the lead furnace." It had recently been said that the slime from Broken Hill, if concentrated by flotation and judiciously mixed with the zinc product, is profitably marketable. It appears, however, that, in negotiations with German and Belgian firms, all sorts of difficulties are raised, and when the sale of concentrated slime without admixture with zinc material is proposed the terms and conditions become impossible. Thus the Broken Hill producers are desirous of trying some alternative process for the purpose of realizing the great accumulations of slime. The Sulphide Corporation is the only Australian producer that is selling slime concentrate to the newly formed firm, operating in Sweden and Norway, the others apparently waiting for the Imbert process (see, for instance, our May issue, p. 259) to become established. This is under option in London to a Broken Hill group that had previously backed the Swedish-Norwegian venture.

Chemical Engineers.

Firing Boilers with Producer-Gas.—*Stahl und Eisen* of Oct. 26, 1911, p. 1771, contains some interesting tests on firing boilers with producer gas, made by the Compagnie des Mines d' Anthrazite de la Mure (Isère) a la Motte d'Aveillans (France). The boiler house contains three boilers with a total heating surface of 4521 sq. ft. Two of these boilers were

double cylindrical with fire tubes of 1615 sq. ft. each and the other was a double cylindrical boiler with water tubes above and of 1292 sq. ft. One fire-tube and one water-tube boiler were used for the test with a total heating surface of 2907 sq. ft. and heated by producer-gas which was generated in a Kerpely producer of 6.9 ft. inside diameter. Fuel was a slaty anthracite, obtained as a waste product from the mines, the size of the coal ranging from 4/5 in. to 2 2/5 in. Two coal analyses may be given:

	I.	II.
H_2O	3.31	2.9
Ash	23.35	22.7
Carbon	71.76	69.4
Vol. Matter	1.58	5.0

Ash analysis:

SiO_2	76.54 per cent
Fe_2O_3	8.84 per cent
Al_2O_3	4.30 per cent
CaO	4.42 per cent
Alcalis	5.90 per cent

Both boilers were filled up to a mark on the gauge glass when started, the test lasted 10 hours and the boilers were then filled up again to the same level. The feed-water temperature was 151 deg. F., and had been used before as cooling water in the water-jacket of the producer. The water was measured in a tank, the steam pressure was kept constant at 80.9 lb. per square inch. The arrangement had already worked for 10 days before the test without any trouble. The total evaporation during the ten hours of the test was 73,193 lb. of water, the coal consumption 11,850 lb., therefore evaporation 6.18 lb. H_2O per pound coal for feed water temperature of 151 deg. F. or 5.5 for water of 32 deg. F. This was very satisfactory since by direct firing this coal evaporated only 2.5 lb. of water per pound of coal. This was due to incomplete combustion on grates, due to its high content of slate. To gasify this coal is advantageous because it permits making use of this coal, of which only the better grades could be sold at all; moreover, the expenses for boiler firing are thereby considerably reduced. One man for a battery of boilers is enough to regulate the gas and air inlet at the burners. Having automatic coal feeders and ash removers, one man can handle two producers with a capacity of 88,184 lb. coal per day. It may be remarked that during the first four hours of the test the producer ran on cool gas with 4/5 per cent CO_2 (Period I). During the following six hours the producer was run somewhat hotter (Period II), causing a poorer gas with 7-8.5 per cent CO_2, but despite this fact the evaporation was higher than before. In the latter case the gas contained more sensible heat. From previous tests it is estimated that the temperature varied between 392 and 482 deg. F. (200 and 250 deg. C.).

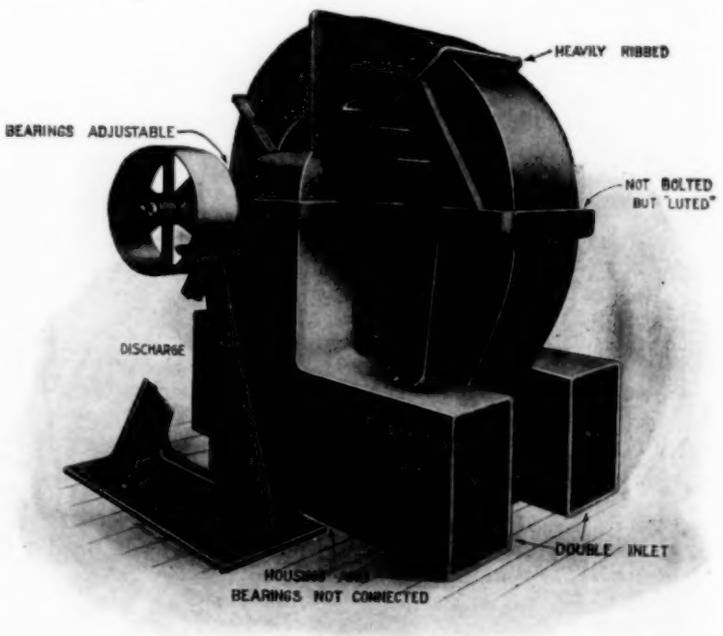
During the first period of 4 hours 26,455 lb. water were evaporated, by 4960 lb. coal, which equals 5.3 lb. H_2O per pound coal; during the second period of 6 hours 46,738 lb. water by 6834 lb. coal, or 6.8 lb. water per pound coal. In other tests similar results were obtained on a slaty hard coal with 35-40 per cent ash, giving an evaporation of 6.5, the producer being run like Period II. In this case the gas main should be as short as possible to reduce radiation losses. For illustration, another example may be cited where a Kerpely producer was directly connected with a heating furnace, the coal was from the Saar district (Germany). In the beginning the producer was operated in the normal way, but since a few years the producer has been run on hotter gas, (but with more CO_2) and it was found that the furnace works better and with a lower coal consumption per ton of production.

Hours.	Boiler pressure in pounds per sq. in.	No. of bags containing 110.2 lb. coal each.	Pounds coal consumption.	No. of tank fillings 881.8 lb. each.	Water evaporated in pounds per hour.	Feed water temperature, degrees Fahr. enheit.	Remarks.
7-8	80.9	12	1322.8	7	6173.	149	Normal gas temp.
8-9	"	12	1322.8	7	6173.	149	"
9-10	"	12	1322.8	8	7054.7	151	"
10-11	"	9	992.1	8	7054.7	156	"
11-12	"	11	1212.5	9	7936.6	154	Running hotter.
12-1	"	11	1212.5	9	7936.6	147	"
1-2	"	9	992.1	9	7936.6	150	"
2-3	"	10	1102.3	8	7054.7	149	"
3-4	"	11	1212.5	9	7936.6	153	"
4-5	"	10	1102.3	9	7936.6	153	"
	80.9	107	11,794.7	83	73,193	151	

Induced Draft for Sulphuric Acid Chambers.

The advantages of fans for producing induced draft in the manufacture of sulphuric acid by the chamber process have long been known to acid manufacturers, but more or less difficulty has been experienced in designing and manufacturing a fan that would meet the requirements.

The principal advantages to be derived from induced draft are:



ACID FAN.

(1) Greater production per cubic foot of chamber volume.

(2) Reduced cost of nitre per pound of acid so produced.

(3) Concentration of a greater amount of chamber stock to acid of 60 deg. Baumé in the Glover tower.

(4) Better absorption of nitre in the Gay Lussac towers on account of the proper excess of oxygen in the chamber process.

(5) Even draft on the sulphur burners.

(6) Successful use of intermediate towers, which are not successfully used where natural draft is used, except in rare instances.

Poor design and construction of fans have been one of the drawbacks to their successful use in induced draft. It would seem to be a simple matter to use a fan for this purpose, but on account of the limiting conditions there have been many

failures. The only material of which a fan for this purpose can be constructed is lead, for the gases are hot and corrosive; but, on the other hand, the great weight of lead and its tendency to distortion have been the cause of failure.

Various designs have been suggested. Some make use of a rigid metal such as iron, covered with lead, but in these the lead has a tendency to become separated from the iron and render the fan inoperative.

More designs have used overhung blast wheels where, on account of the necessary thickness and weight of the lead, the wheel has to be balanced perfectly and the journal boxes adjusted to a fine point. If these conditions are not observed, the blast wheel, projecting beyond the bearings into the fan housing, would revolve eccentrically; the great weight would bend the shaft, and the blades striking the housing would be broken; or the metal in the blades would become crystallized from vibrations and break. Considering the speed at which these fans must be operated, the possibility of breaking the blades becomes a source of danger as well as trouble and expense. Fans of this type also become unbalanced due to corrosion.

In order to overcome these difficulties and objections in acid fans, a design shown in the accompanying illustration has been made by Mr. N. L. Heinz and Mr. L. B. Skinner, who are practically engaged in acid manufacture. The machine is made and marketed by the Stearns-Roger Manufacturing Company, Denver, Col. It probably is more expensive to manufacture than other designs, but, when its service is considered, the increased cost is more than counterbalanced by continuous operation without constant attention and frequent repairs.

The lead alloy of which the fan and housing are made is of the proper mixture to resist acid corrosion, and the tensile strength of the metal is not impaired by improper handling. The blades are of the right number and shape to give the highest efficiency, fitting closely to the housing. The latter is heavily ribbed to give stability, and avoid the necessity of constant adjustment due to warping. The bearings of the fan are separate from the lead housing, thus reducing the possibility of tearing the latter by vibration. Accessibility is provided by luting the cover of the housing to the base, instead of bolting the two parts together.

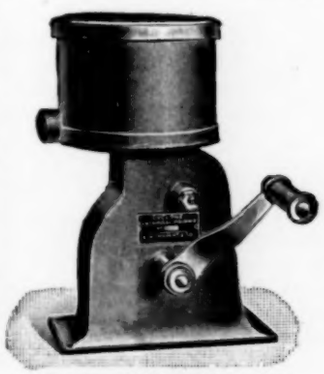


FIG. 1.—HAND-DRIVEN 5-IN. CENTRIFUGAL.

The general construction of this machine is such as to make the high rate of speed of the fan absolutely safe as far as possibility of accident is concerned. It operates at 475 r.p.m., and will furnish the gas necessary for the operation of a chamber set of 300,000 cu. ft. with intermediate towers. The approximate shipping weight of the machine is 7500 lb.

Centrifugals for Laboratory Work.

One of the quickest and most efficient methods to remove crystals and precipitates from mother-liquors and solutions is undoubtedly to use centrifugal force.

This fact is amply demonstrated by the enormous amount of large centrifugals in use in many industries. In the laboratories of chemical and pharmaceutical factories, sugar works, dye-houses, technical schools and metallurgical industries, the only difficulty which has heretofore prevented the use of smaller centrifugals has been their relatively high price and bulkiness as they were generally the reduced models of the big machines.

In the accompanying illustrations, however, is shown a centrifugal which, on account of its compactness and extreme simplicity, can be produced at a much lower cost than has heretofore been possible.

The Chavant "cyclone" laboratory centrifugal has been on the market for many years. It was formerly made in Jersey

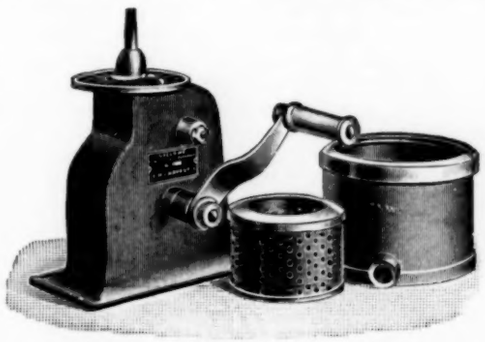


FIG. 2.—BASKET AND CASING REMOVED.

City, N. J., but the works were recently moved to Fontenay-sous-Bois, France, where a larger plant and better manufacturing facilities were to be had with the enormous European field near at hand.

The design and construction of the "cyclone" centrifugal have been improved in many respects. The new model, known as

FIG. 3.—HAND-DRIVEN 8-IN. CENTRIFUGAL.

Type "F," is made in three sizes: 4½, 5 and 8 in., and is furnished with a crank for hand-drive or a pulley for power-drive. The machine, mounted on ball-bearings, is exceptionally light-running and can be operated at high speed without any perceptible vibration.

Fig. 1 shows type F, 4½ and 5-in., hand-driven. In Fig. 2

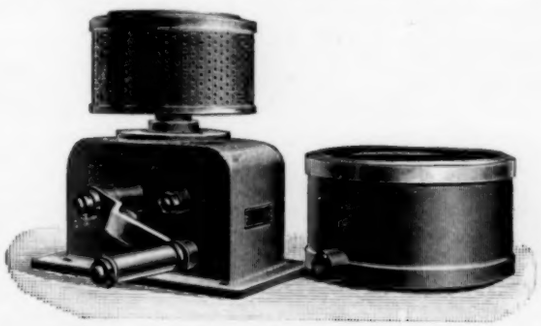


FIG. 4.—CASING REMOVED.

basket and casing are removed. Fig. 3 shows type B, 8-in., hand-driven, while Fig. 4 shows the same with casing removed.

There are no pulleys or belts in the internal mechanism, which consists of machine-cut gears, made with the greatest accuracy by special machinery in order to insure long wear and easy running.

The gears, lubrified by outside oilers, are encased in the base casting fully protected from dust, but can be easily reached for cleaning as no part has to be unscrewed.

The basket and also its casing are detached by simply lifting them off the machine and they can be replaced instantly.

The basket is made of bronze and brass and is perfectly balanced. For acid products it is tinned and its casing lined with porcelain. A hard rubber basket can be supplied too.

For all "cyclone" centrifugals the manufacturer guarantees that with a maximum of 70 revolutions of the crank or of the pulley per minute they will give all the results which shall be expected from a first-class laboratory centrifugal.

The general sales agent of the company is Mr. Victor J. Boulin, with offices at 123 Liberty Street, New York City.

Repair of a Large Rock-Crusher Shaft.

A repair of exceptional size was recently executed for the North River Stone Company, of Kingston, N. Y., on one of its rock-crusher shafts. The shaft in question was 18 ft. long and ranged from 12 in. to 22 in. in diameter. It was 14 in. in diameter at the point of fracture.

It was decided to make the repair by the thermit process, but owing to the fact that there were not adequate facilities at Kingston for aligning the pieces accurately, they were sent to the Jersey City shops of the Goldschmidt Thermit Company, where they could be placed in V blocks, mounted on a perfectly level bed plate and so lined up accurately, previous to welding them together.

There is no necessity to describe here again the well known thermit reaction of Dr. Hans Goldschmidt between aluminium and iron oxide, yielding a slag of alumina and as chief product a molten highly superheated low-carbon steel, available for welding. By mixing steel punchings with the thermit the carbon content of the thermit steel is increased, and the quality of the steel is also improved by the addition of other materials such as manganese, nickel or chromium.

The temperature attained by the reaction is approximately

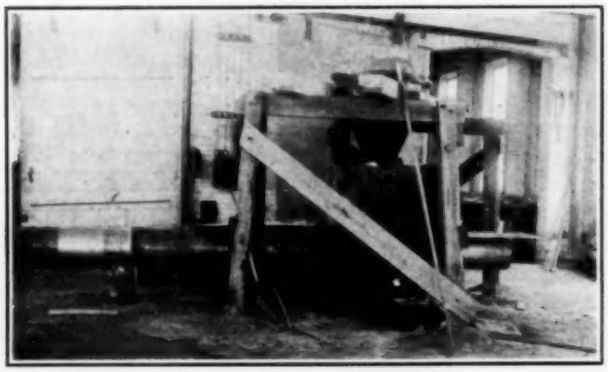


FIG. 1.—PREHEATING SHAFT PREVIOUS TO WELDING BOX AND CRUCIBLE IN POSITION.

5400 deg. Fahr. and it will readily be seen that by pouring this superheated steel into a mold surrounding the ends of the sections to be welded together it will dissolve the metal with which it comes in contact and amalgamate with it to form a single homogeneous mass when cool.

In welding the large shaft for the North River Stone Company a space of 2¼ in. was provided between the ends of the sections to be united by cutting away the broken faces to this extent, after which a pattern of yellow wax was formed about the sections in the shape of a collar 14 in. wide and 2 in. thick at the middle point. In cross-section this collar resembled in shape a segment of a circle. A sheet-iron mold box was then placed in position and rammed with special molding material, consisting of a mixture of one-third fire clay, one-third ground fire brick and one-third fire sand. Ordinary molding material would not be sufficiently refractory to withstand the high temperature of the thermit steel.

Wooden patterns were used for two pouring gates, a riser directly over the top of the weld, and a small preheating opening at the bottom. The flame of a powerful gasoline compressed air torch was then directed into the preheating hole and melted out the wax, thus leaving the mold ready for pouring.

The preheating, however, was continued for a period of ten

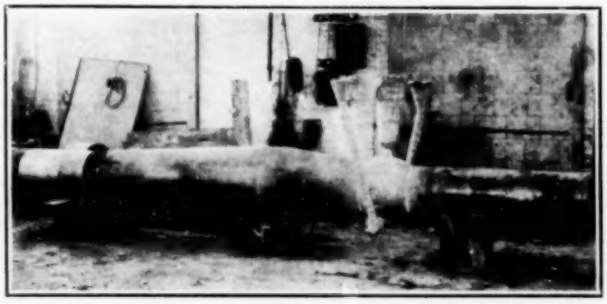


FIG. 2.—VIEW AFTER WELD IS FINISHED, BUT BEFORE CUTTING OFF METAL LEFT IN POURING GATES AND RISERS.

hours so as to thoroughly bake the mold and bring the sections to be welded together to a bright red heat, at which time the torch was withdrawn, the preheating hole closed up and the thermit charge, held in two crucibles suspended over the pouring gates of the mold, ignited.

The thermit charge for this repair consisted of 1100 lb. of thermit, 275 lb. of steel punchings, 11 lb. pure carbonless manganese and 11 lb. of chromium thermit. These were all thoroughly mixed with the thermit in the crucibles.

The thermit reaction required about 35 seconds for completion, but the charge was held in the crucibles a few seconds longer so as to allow time for the slag to thoroughly separate from the steel. The slag, being lighter than the steel, floats to the top of the crucibles, while the steel sinks to the bottom. The crucibles are tapped from the bottom, so that the steel flows into the mold first, being led to the lowest point of the mold, then up between and around the sections to be welded into the riser. The slag does not enter the mold at all, but merely overflows.

It is important that the metal in the weld be allowed to cool slowly and in the case of this 14-ft. shaft the mold was left undisturbed for a period of 45 hours, after which it was broken up and the steel left in the riser and pouring gates removed by means of a cutting flame of oxygen and illuminating gas.

The entire repair was carried out in less than a week, including the time required for towing the shaft on a barge from Kingston to New York and back again. This alone took over two days.

The thermit steel collar or reinforcement was not machined off, as it did not interfere with the operation of the shaft and added considerably to the strength of the repair.

If the thermit process had not been available for this work, the owners would have been forced to wait several weeks for a new shaft, while the cost of a new shaft alone would have amounted to several times the cost of the weld.

Mr. Paul O. Abbé announces the opening of an office at room 1201, Johnston Building, 30 Broad Street, New York City, as a builder of jar and pebble mills in all styles and sizes and as agent for the sale of mixers, sifters, vacuum pumps, pressure blowers, and bolting cloth of various makes.

Frictionless Ink-Type Recording Instrument.

An illustrated description of Bristol's patented semi-transparent smoked chart recorder was printed in our issue of January, 1908 (Vol. VI, p. 35). Although these recorders are fundamentally simple in their construction and hundreds are in daily service, there has been a call for a recorder in which the record is made with ink. To meet the demand for a frictionless ink-type recording instrument which would parallel

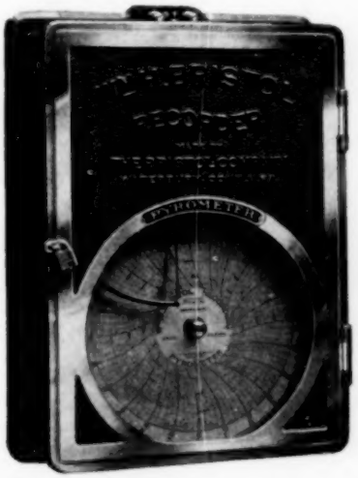


FIG. 1.—OUTSIDE VIEW OF RECORDER.

the smoked-chart recorder as nearly as possible in intrinsic simplicity, and accurately record fractions of millivolts and be adapted for use as a recording electric pyrometer, the instrument illustrated herewith has been developed and placed upon the market by the Bristol Company, of Waterbury, Conn.

These instruments have been thoroughly tested out in practical service for two years past, and are the result of several years of study and experience with an original design patented

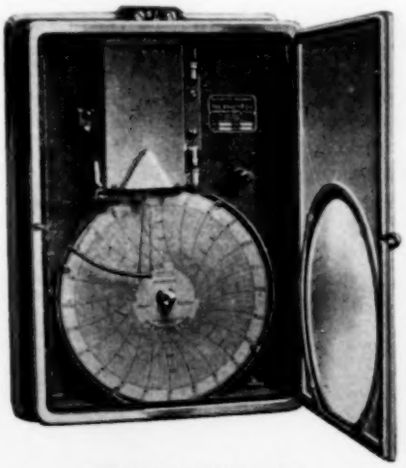


FIG. 2.—INTERIOR VIEW.

by Prof. Wm. H. Bristol on April 13, 1909, of a frictionless ink recorder using a hinged electrical movement carrying a retaining receptacle for marking fluid which extends over the path of the recording tip, and is provided with means for periodically making contact with the source of marking fluid and the chart.

Fig. 1 shows the recorder ready for operating.

Fig. 2 is an interior view showing the galvanometer movement case hinged to the back of the instrument and carrying the inking pad in front of the recording arm.

Fig. 3 shows the sensitive electrical movement swung to one side for convenience in removing the record and inserting a fresh chart. The capillary gold tube open at both ends is carried at the end of the recording arm at right angles to the surface of the chart. The inking pad is suspended from the case of the electrical movement and is curved to correspond with the arc covered by the motion of the end of the recording arm.

When the movement is swung back into its operating position, as shown in Fig. 2, the recording arm can swing free, accommodating itself to the position corresponding to the delicate current which is to be measured. The clock which revolves the chart at the desired speed also automatically presses the inking pad toward the chart every 10 seconds, bring-

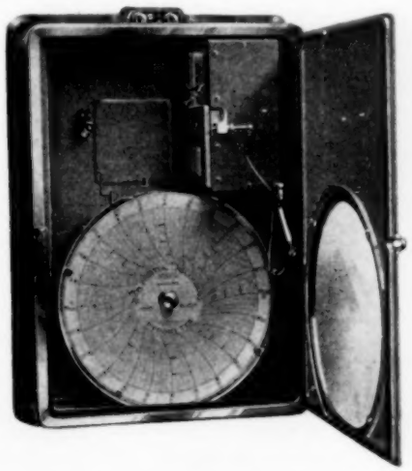


FIG. 3.—INTERIOR VIEW.

ing one end of the capillary tube into contact with the chart and the opposite end simultaneously into contact with the inking pad. A fine dot of ink is left on the chart and the capillary tube is replenished with ink from the pad. The recording arm thus carries a constant supply of ink, and its perfect balance, which is very important, is always maintained. The electrical movements used in these recorders are made especially for this purpose by the Weston Electrical Instrument Company.

Although the most important applications of these recording instruments have been for pyrometers, they have also been used for electrolytic research, recording voltmeters and recording shunt ammeters.

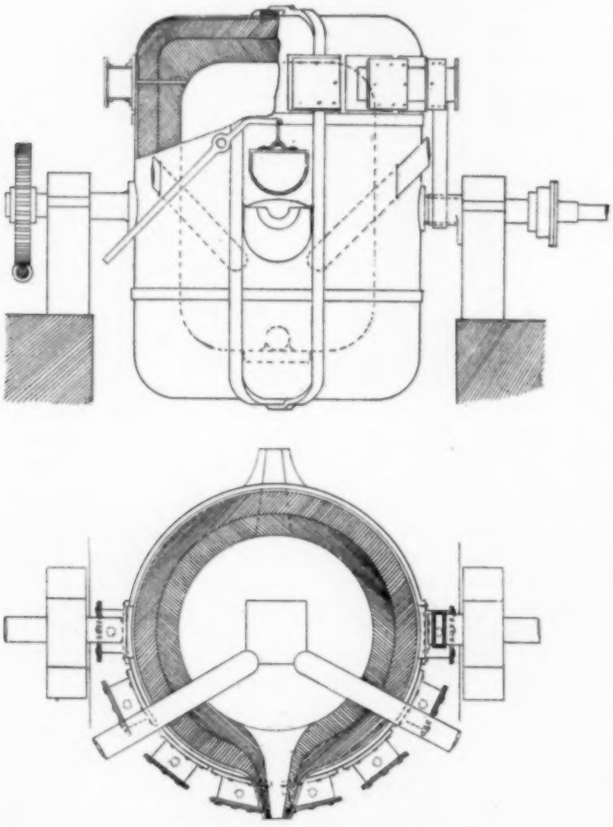
In the determination of sulphur in ores difficulty is frequently encountered in getting all the sulphur in solution before it separates out and collects in a globule of elementary sulphur. When this condition occurs it necessitates longer treatment to dissolve the sulphur. In order to overcome this obstacle, E. H. Westling proceeds as follows: The solvent used is nitric acid of 1.35 sp. gr. with a little dissolved potassium chlorate, at a temperature of about 60°C. The boiling point of this mixture is below the melting point of sulphur, and if the ore pulp is added gradually to the solution sulphur will not separate out but will be dissolved without difficulty. Rapid and complete decomposition is claimed for this procedure.

The Blum Zinc Oxide Process, which is protected by U. S. patent 993,019, consists in heating a charge made up of alternate layers of fuel and mixed zinc ore and fuel. It has been customary to carry out this process on a perforated grate, after the manner of the Wetherill process. Mr. Blum, however, proposes to use a pot like those used in the Huntington-Heberlein roasting system. The layers of alternate ore and fuel are 10 cm thick. The fuel is anthracite fine, both for mixing with the ore layer and for the fuel layer. The advantage claimed is large capacity as compared with Wetherill grates.

Electro-Bessemer Furnace.

In this issue, in the department of Recent Metallurgical and Chemical Patents, an interesting patent of Messrs. Walker and Gray, of the U. S. Steel Corporation, is mentioned, describing a steel refining furnace, which is first operated as an open-hearth and then as an electric furnace, thus obviating the necessity of transfer of the molten metal from one furnace to another.

The analogous idea for Bessemer converter practice is incorporated in the Electro-Bessemer furnace, designed by Messrs. Verdon Cutts & Hoult, 43 Edge Hill Road, Sheffield, England. This furnace was recently described in a paper by Mr. Verdon O. Cutts before the Staffordshire Iron and Steel Institute. After reviewing the different types of electric furnaces in general and those for iron ore reduction and steel refining in particular, Mr. Cutts described the Electro-Bessemer furnace as follows:



ELECTRO-BESSEMER FURNACE.

"The primary object which we had in view in designing this furnace was the reduction of working costs. We recognized the fact that the cost of electrical energy very considerably affects the cost of steel manufacture in the electric furnace. Our experience in important steel works both in this country and on the Continent has revealed a very high cost of generating electrical energy, and, in my opinion, this has proved sufficient at least to delay the adoption of the electric furnace in many steel works. It was, therefore, our object to produce a furnace in which electrical energy is only used where it offers unique advantages over other methods. A careful analysis of the subject shows that the value of the electrical process lies not so much in the making of the steel as in the finishing of the same. This has also been recognized by Héroult, Girod, and others, and accounts for their furnaces being now largely employed for the treatment of steel made in the ordinary Bessemer converter or open-hearth furnace. By our process the steel is made by the usual Bessemer method and finished electrically without it being necessary to pour the molten metal from one vessel to another during the refining operation. The loss of heat and expenditure of labor which are inseparably connected with the duplex process above referred to are thus avoided in the Electro-Bessemer furnace, resulting in a comparatively very high efficiency.

"Further, the Electro-Bessemer furnace is less dependent upon skilled manipulation than is the alternative combination of separate Bessemer and electric furnaces. . . .

"The many complicated designs which have been originated during the past few years have proved conclusively that the safest and shortest way to practical success is the utmost simplicity in construction, and the incorporation of old, well-known and successful methods with new processes. This we have kept well in mind, and the Electro-Bessemer process, as already suggested, combines in principle the ordinary Bessemer furnace and the electrical refining process. The result is that steel is produced cheaply and quickly as in the Bessemer furnace, and the disadvantage of lack of uniformity in steel so produced is entirely overcome by applying the electrical process of final refining.

"The Electro-Bessemer furnace is a closed vessel, having an opening in one side, through which the charge may be introduced and slags added and withdrawn. It is mounted on bearings, and is capable of being tilted within an angle of rather more than 180 degrees. One end of the furnace is fitted with blast tuyeres for blowing. At the other end of the vessel electrodes are arranged, entering through the furnace wall and being adjustable by any of the ordinary independent methods, either electrical or mechanical. A tapping hole is provided at this end of the furnace, which may be used for the removal of the finished product, if desired. Samplings may be taken from the charging opening, which opening might also be used, if desired, for pouring molten metal. The electrodes are fitted with cooling jackets of cold air or water.

"The process is simple in the extreme, being briefly as follows. The furnace is preheated in the ordinary way, and is then charged with molten metal direct from the cupola or blast-furnace mixer, the furnace being tilted into a horizontal position. It is then brought into a vertical position, the "Bessemer" portion containing the charge. The blast comes into action automatically as this position is gained, and the process of conversion proceeds as in the ordinary Bessemer converter. Meantime the "electric" portion of the furnace is heated by the hot gases, etc., resulting from the process, and for the ultimate removal of which gases ample provision is made. On the completion of this part of the process the furnace is tilted and the metal transferred to that end which is provided with electrodes. Current is switched on, and the refining operation continued to any desired extent. Both arc and resistance heating are obtained in the furnace.

"The Electro-Bessemer furnace may be either basic or acid lined; if it is basic lined it is possible to start with impure basic Bessemer pig-iron and to produce finished steel or better analysis than the best acid open-hearth steel. If the furnace has an acid lining, chemical refining of the blown metal does not take place, but the killing of the metal by heating in the 'electric' end of the furnace results in the production of particularly sound castings and ingots free from blow-holes.

"The utilization of the flame from the Bessemer blowing for the heating of the 'electric' hearth results in a considerable saving of electrical energy during the second stage.

"Having poured away the slag containing the bulk of the impurities at the end of the Bessemer blow (assuming a basic lining), new, clean, purifying slags can be added.

"Assuming that the phosphorus content is already low when the electrical operation commences, the energy consumption is less than 100 kw. hours per ton of steel produced. If, however, both desulphurization and dephosphorization are included, about 190 kw. hours are consumed."

Automatic Controllers for Motor-Driven Machinery.

The Electric Controller & Manufacturing Company, of Cleveland, O., has recently placed on the market a line of automatic controllers designed for the specific purpose of giving the utmost convenience in the control of motor-driven machinery.

It has been estimated that the output of many machines can

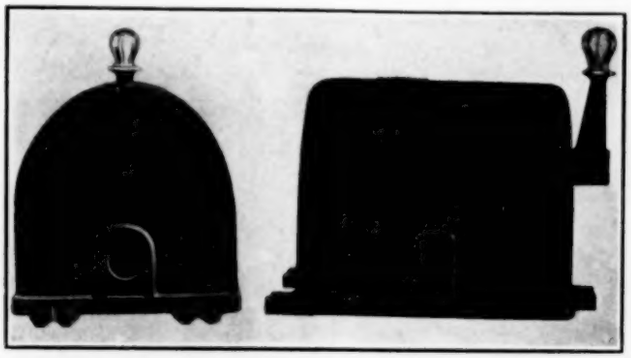


FIG. 1.—FRONT VIEW AND SIDE VIEW OF OPERATOR'S SWITCH.

be increased 20 per cent by the central grouping and convenient arrangement of all the operating levers. "Handiness of control" is recognized as being very important in securing the utmost production from a machine, and the automatic controllers described in this article were designed to provide this "handiness of control" for starting, stopping or reversing the motor and machine.

FIG. 2.—ACCELERATING UNIT.

The controller consists of a small operator's switch, shown in Fig. 1, and an accelerating unit which automatically accelerates the motor during starting and retards it during stopping. This accelerating unit is built in four different types, two of which are shown in Figs. 2 and 3.

The controllers are built in three types to secure: First, non-reversing and electric braking; second, reversing without electric braking; third, reversing and electric braking. In each type a variety of four different forms of accelerating units are

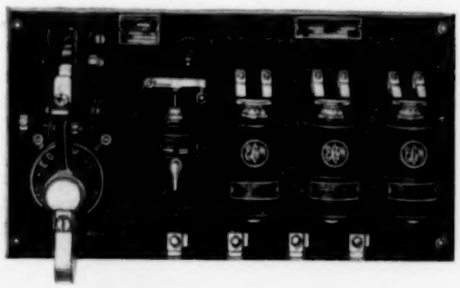


FIG. 3.—ACCELERATING UNIT.

offered. These accelerating units vary in their design from a simple train of accelerating switches up to a unit having a fused service switch, a train of accelerating switches, and complete circuit-breaker features.

The accelerating unit automatically accelerates or retards the motor through the action of series-wound accelerating switches which possess the remarkable characteristic of acting, not only as switches, but as current limit relays as well. When the current in the winding of one of these switches exceeds a predetermined value the switch locks open and cannot close until the current is reduced to the proper value.

When the operator's switch is thrown to the running position, current flows through the motor, through all of the starting resistance, and through the coil of the first series wound accelerating switch. As the motor accelerates the current drops, and when it reaches the correct value the first accelerating switch closes, cutting out a portion of the starting resistance. The succeeding accelerating switches operate similarly, ultimately cutting out all of the starting resistance and putting the motor across the line. By throwing the operator's switch to its original position, the motor is disconnected from the line and consequently stops. Different positions of the handle of

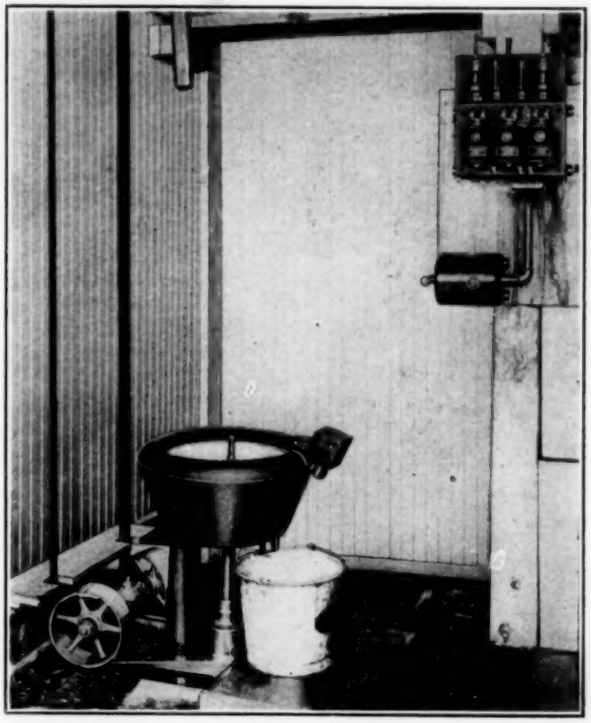


FIG. 4.—MOTOR-DRIVEN CENTRIFUGAL OIL EXTRACTOR EQUIPPED WITH AUTOMATIC CONTROLLER.

the different types of operator's switch provide for drifting, reversing, or rapid stopping by dynamic braking.

Electric braking is secured by a change of connections, accomplished by the operator's switch, which first inserts all the starting resistance in series with the armature. The motor is then quickly and evenly brought to rest by automatic electric braking, the accelerating switches, in this case, acting as retarding switches by cutting out, step by step, the resistance as the current, generated by the motor, decreases due to the slowing down of the motor.

The manufacturers claim the following important advantages for this automatic controller:

It limits the acceleration and retardation current at all times to a safe value.

It accelerates and retards the motor in the minimum safe amount of time, and automatically varies the time of acceleration and retardation, depending upon the load which the motor has to start and stop.

It provides the best conditions for good commutation.

By limiting the current, both in starting and stopping, it limits all mechanical strains on the motor and driven machinery.

It obviates the necessity of mechanical clutches on many motor-driven machine tools.

It adds very materially to the safety of an installation, since in case of accident the motor may be quickly stopped.

It inherently provides no-voltage protection, for if the voltage fails the switches drop open, and upon the return of voltage they automatically close in their regular method and sequence, again accelerating the motor to full speed.

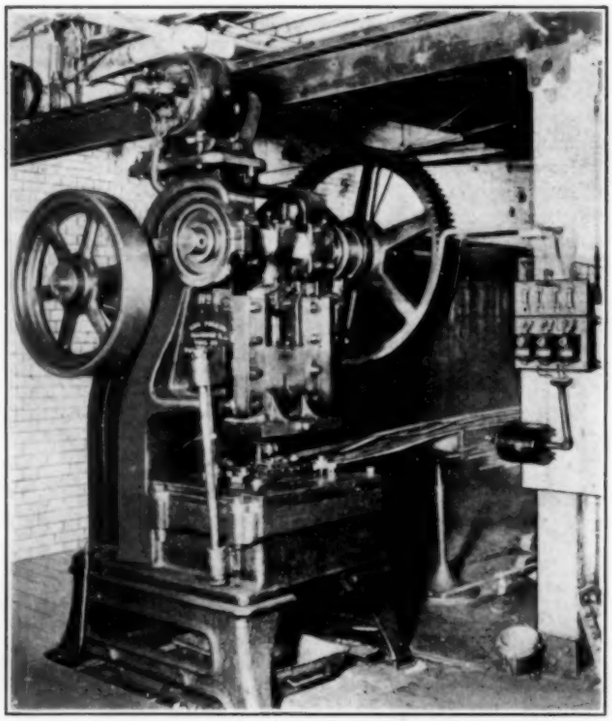


FIG. 5.—MOTOR DRIVEN PUNCH PRESS EQUIPPED WITH A TYPE H. N. B. FORM A. K. AUTOMATIC CONTROLLER.

Fig. 4 shows a motor-driven centrifugal oil extractor equipped with a type H N B, from A K automatic controller, Fig. 5 a motor-driven punch press equipped with the same type of controller. In both illustrations the automatic controller can be clearly seen on the right of the picture, the operator's switch being placed right below the accelerating units.

Census Report on Sulphuric, Nitric and Mixed Acids.

A preliminary statement of the general results of the thirteenth census of establishments engaged in the manufacture of sulphuric, nitric and mixed acids has just been issued by Director Durand, of the Bureau of the Census in the Department of Commerce and Labor. It was prepared under the direction of William M. Steuart, chief statistician for manufactures, Bureau of the Census. The figures are subject to such revision as may be necessary after a further examination of the original reports.

There were 42 establishments engaged in this industry in 1909 and 32 in 1904, an increase of 31 per cent.

The capital invested as reported in 1909 was $18,726,000, a gain of $5,964,000, or 47 per cent, over $12,762,000 in 1904. The average capital per establishment was approximately $446,000 in 1909 and $399,000 in 1904.

The value of products was $9,884,000 in 1909 and $9,053,000 in 1904, an increase of $831,000, or 9 per cent. The average per establishment was approximately $235,000 in 1909 and $283,000 in 1904, a decrease of 6 per cent.

The cost of materials used was $5,386,000 in 1909, as against $4,973,000 in 1904, an increase of $413,000, or 8 per cent. In addition to the component materials which enter into the products of the establishment for the census year there are included fuel, rent of power and heat, and mill supplies.

The value added by manufacture was $4,498,000 in 1909 and $4,080,000 in 1904, an increase of $418,000, or 10 per cent. This item formed 46 per cent. of the total value of products in 1909 and 45 per cent. in 1904. The value added by manufacture represents the difference between the cost of materials used and the value of products after the manufacturing processes have been expended upon them. It is the best measure of the relative importance of industries.

The miscellaneous expenses amounted to $916,000 in 1909 and $713,000 in 1904, an increase of $203,000, or 28 per cent. Miscellaneous expenses include rent of factory or works, taxes and amount paid for contract work, as well as such office and other expenses as cannot be elsewhere classified.

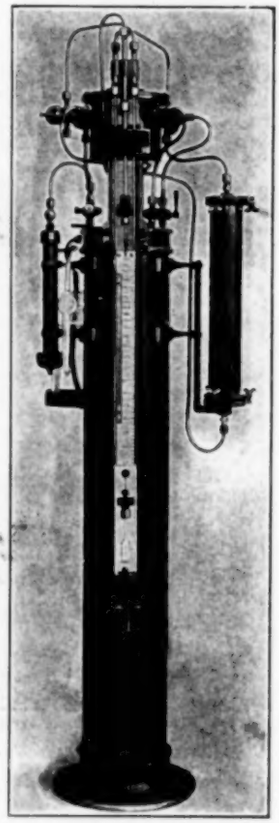


FIG. 1.—CONTINUOUS CARBON DIOXIDE AND TEMPERATURE RECORDER.

The salaries and wages amounted to $2,046,000 in 1909 and $2,062,000 in 1904, a decrease of $16,000, or 1 per cent.

The number of salaried officials and clerks was 330 in 1909 and 308 in 1904, an increase of 7 per cent; their salaries decreased from $556,000 in 1904 to $551,000 in 1909, or 1 per cent. The average yearly salary per official or clerk, therefore, decreased from $1,805 in 1904 to $1,670 in 1909.

The average number of wage earners employed during the year was 2252 in 1909 and 2447 in 1904, a decrease of 8 per cent; their wages decreased from $1,506,000 in 1904 to $1,495,000 in 1909, or 1 per cent. The average yearly wages per man, therefore, increased from $615 in 1904 to $664 in 1909.

The primary horse-power was 6494 in 1909 and 5416 in 1904, an increase of 20 per cent.

The average horse-power per establishment, considering all establishments, was approximately 155 horse-power in 1909 and 169 in 1904.

The total production of sulphuric acid, reduced to 50 deg. Baume, was 2,797,723 tons in 1909 and 1,869,446 in 1904, an increase of 50 per cent. Of this total $933,848 tons were reported in 1909 and 467,614 in 1904 by establishments making it as a chief product, an increase of 100 per cent,; 621,801 tons in 1909 and 433,377 in 1904 by establishments engaged primarily in the manufacture of other products, and 1,242,074 tons in 1909 and 968,455 in 1904 were made and consumed in establishments where manufactured.

The total production of nitric acid was 138,086,565 pounds in 1909 and 108,380,387 in 1904, an increase of 27 per cent. Of this total 8,396,326 pounds were reported in 1909 and 30,306,555 in 1904 by establishments making it as a chief product, a decrease of 72 per cent.; 18,929,620 pounds in 1909 and 15,927,526 in 1904 by establishment engaged primarily in the manufacture of other products; and 110,760,619 pounds in 1909 and 62,146,306 in 1904 were made and consumed in establishments where manufactured.

The total quantity of mixed acid manufactured in 1909 cannot

be shown because the amount made and consumed in establishments where manufactured was not reported; but the quantity manufactured by establishments making it as a chief product and by those reporting it as a by-product was 57,182,168 pounds in 1909, compared with 65,331,327 in 1904, a decrease of 12 per cent.

The production of establishments making mixed acid as a chief product was 45,361,626 pounds in 1909 and 42,812,894 in 1904, an increase of 6 per cent.

A comparative summary follows, giving the general statistics for the industry in 1904 and 1909:

SULPHURIC, NITRIC AND MIXED ACIDS—GENERAL SUMMARY, 1909 AND 1904.

	—Census.—		Per cent. of increase,
	1909	1904.	1904-1909.
Number of establishments..	42	32	+ 31
Capital	$18,726,000	$12,762,000	+ 47
Cost of materials.....	5,386,000	4,973,000	+ 8
Salaries and wages.....	2,046,000	2,062,000	— 1
Salaries	551,000	556,000	— 1
Wages	1,495,000	1,506,000	— 1
Miscellaneous expenses....	916,000	713,000	+ 28
Value of products.....	9,884,000	9,053,000	+ 9
Value added by manufacture (products less cost of materials).....	4,498,000	4,080,000	+ 10
Employees:			
Number of salaried officials and clerks.....	330	308	+ 7
Average number of wage earners employed during the year.....	2,252	2,447	— 8
Primary horse-power.....	6,494	5,416	+ 20

Continuous Carbon Dioxide and Temperature Recorder.

In connection with the paper of Mr. Edward A. Uehling, printed elsewhere in this issue, a description of the Uehling continuous carbon-dioxide and temperature recorder should be of particular interest. It is made by the Uehling Instrument Company, of Passaic, N. J., and consists of a carbon dioxide and temperature measuring machine as shown in Fig. 1, to which are connected by small copper tubing standard type M Uehling pressure and vacuum recorders, properly calibrated for CO_2 or temperature.

Indicators of the standard type Z construction, which are based on the elementary U-tube principle may of course be connected to the machine in addition to the recorders. Thus, for instance, a standard equipment for a boiler plant would consist of the carbon dioxide and temperature machine shown in Fig. 1, which has at the front a CO_2 and a temperature indicator, and in addition to this machine an additional indicator located at the boiler front, and two type M recorders located in the engineer's office, one calibrated for CO_2, and the other for temperature.

Another style machine is shown in Fig. 2. This is a double combined machine, making two measurements of CO_2 and two measurements of temperature. It will be noted that at the front of the machine there are two indicators for temperature and two for CO_2. The full equipment comprises two other indicators placed at the front of the boilers, for the CO_2, and four standard recorders, two calibrated for temperature and two calibrated for carbon dioxide.

The two styles of machines illustrated and the two equipments above described are typical of the numerous arrangements which are possible when a continuous carbon dioxide and temperature machine of this type is used. Standard-style machines include those giving four measurements of CO_2 and no temperature measurements, or again, three measurements of CO_2 and one of temperature, or two measurements of CO_2, or else simply one measurement of CO_2. In every case the equipment can consist of as many recorders and indicators as desired, although the standard equipment comprises extra indicators at the boiler front for CO_2 and one standard recorder for each measurement of CO_2 and temperature.

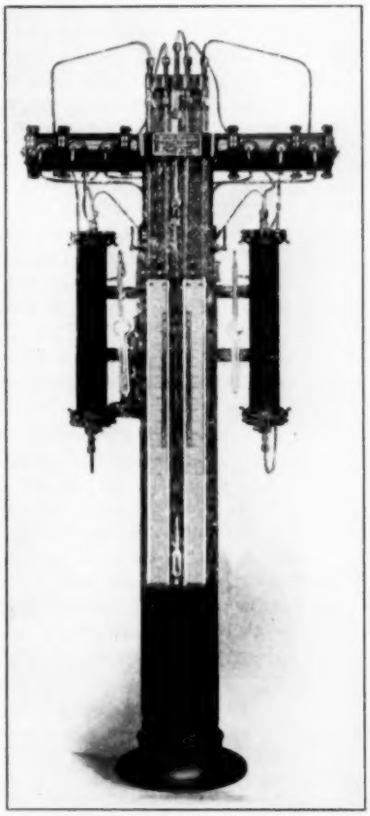


FIG. 2.—DOUBLE CO_2 AND TEMPERATURE RECORDER

This flexibility of equipment as also the ability to use standard recorders and indicators, and to obtain continuous measurement, all follow from the fact that the Uehling continuous carbon dioxide temperature machine operates on the pneumatic principle. By suction apparatus in the machines, a constant vacuum is maintained which draws the gas to be analyzed for CO_2 through two apertures. Between the two apertures, the CO_2 contents of the gas are removed by caustic potash. The more CO_2 contained in the gas, the more is removed and the less gas has to pass through the last aperture, and therefore the higher will be the vacuum between the two apertures, so that the vacuum or suction in this chamber becomes a measure of the amount of CO_2 contained in the sample of gas drawn from the boiler. A vacuum indicator and recorder connected to this chamber will measure the per cent CO_2, and readings can be obtained if the CO_2 chart is properly calibrated.

The same principle is applied to measuring the temperature. Here, however, the two apertures through which the gas, or air, as it is in this case, passes, are not kept at the same temperature. In applying the pneumatic principle to the measurement of temperature, one aperture is located in a nickel tube which is placed in the hot gas whose temperature is to be measured, while the other aperture is kept at a constant lower temperature. As the air has a higher temperature when passing through the first aperture than through the second, the suction or vacuum in the chamber between these apertures will increase with increase of temperature, because the volume of air varies directly with the temperature (absolute). Therefore, the vacuum between the apertures needs only to be transmitted to the indicators or recorders by small copper tubing, in order to obtain any number of accurate and reliable measurements.

Water Power Withdrawals.—Over a million and a half acres of public land now stand withdrawn by the Government as being valuable for the possible development of water power. During October the United States Geological Survey recommended the withdrawal of 54,422 acres as having power possibilities and the restoral of 2600 acres in Montana and Wyoming because field examination by the Survey showed them not to be valuable for that purpose.

Welding.

At a meeting of the American Society of Mechanical Engineers held in New York, on Nov. 14, the general subject of autogenous welding was discussed in great detail. Mr. H. R. Cobleigh presented an introductory paper dealing with the general aspects of the subject, in which were described the origin and principles of each process and the apparatus used in each. He outlined the features of electric resistance welding, electric arc welding, thermit welding, oxy-acetylene welding and oxy-hydrogen welding. By means of numerous lantern slides the mechanical details of the apparatus employed in each system of welding were described. It was stated that each method possesses certain advantages and certain disadvantages, so that each system may be considered most advantageous for certain particular services. The oxy-acetylene flame produces a higher temperature than oxy-hydrogen, but operates under the disadvantage of a marked tendency to flash back. This tendency has been overcome by the use of special forms of torches, which require the gases to pass through porous material before being ignited. The oxy-hydrogen apparatus has a less tendency to flash back, but the temperature is not so high. The oxygen and hydrogen for the oxy-hydrogen apparatus can conveniently be produced by electrolytic dissociation of water. The acetylene used in the oxy-acetylene torch is produced from calcium carbide, according to the methods employed with illuminating acetylene. The acetylene is produced by dropping water upon the carbide, or by dropping the carbide into water, the latter method having certain advantages. The oxygen used in the oxy-acetylene flame can be produced either by chemical or electrolytic processes. In any event, it is most advantageous to convey the gases in highly compressed form.

Mr. G. E. Pelissier described the Goldschmidt thermit process of welding, which depends upon the chemical combination of aluminium with oxide of iron, and the resultant products being oxide of aluminium and iron. By the use of motion pictures the thermit welding of rails was shown in a striking and most interesting manner. This process has found numerous applications in heavy engineering, such as that in connection with locomotive and ship machinery repairing.

Mr. C. B. Auel gave much interesting information relating to the Bernardos, Slavianoff and Zerener processes of electric welding. These processes differ according to the material used for electrodes. In the first-named process one electrode is formed by the metal to be welded, and the other electrode is of copper. In the second process the two metallic parts to be joined form the electrodes, while in the third process there are two carbon electrodes, with the metal forming an intermediate contact between these two electrodes. Much attention was devoted by the author to the Bernardos process, which he considered to be the most universally applicable. However, the Slavianoff process produces a stronger joint. In this process use is made of an arc of a length of about ¼ in., an emf force of 25 volts and a current of 150 amp. In the Bernardos process the length of the arc would be about 1 in., the current from 160 to 700 amp and the emf from 45 to 50 volts at the arc.

Mr. W. H. Brown described a welder acting according to the Zerener process, which he has found highly advantageous on account of the economy in operation, the consumption being 18 kw as compared with 50 kw by other electrical processes.

The subject was discussed by Messrs. W. H. Spire, J. D. Mooney, Henry Cave, W. R. Noxon, W. H. Lavin, W. J. Fritz, J. F. Springer, E. B. Katte and Dr. Lieber.

Mr. Noxon called attention to the stoppage of leaks by means of oxy-acetylene welding in a pipe line in central Colorado, which carries water under a head of 1875 ft. The cost of welding this pipe line was only a small fraction of the cost which would have been involved in replacing the pipe line.

Mr. Lavin stated that the oxy-hydrogen process has been widely adopted for regular use in industrial establishments for welding purposes. He claimed as an advantageous feature the fact that the oxygen and hydrogen gases for the flame can be obtained in a high degree of purity by the electrolytic method.

Mr. Fritz showed numerous illustrations of the oxy-acetylene flame, which is now being employed for dismantling the ill-fated battleship Maine in the Havana harbor.

Mr. Springer attributed the high temperature of the oxy-acetylene flame to the concentration of heat units in the small space.

Dr. Lieber stated that the Blau gas which is being used in welding is compressed and liquified until it occupies only 1/400 part of its original volume. As compared with acetylene having 1500 heat units per cu. ft., the Blau gas has 1853 heat units.

New Detinning Works.

The Goldschmidt Detinning Company, of New York City, is erecting a new detinning plant in East Chicago, Ind., with a capacity of about 40,000 tons of tin scrap per year. This plant will have several novel and very interesting features.

Instead of using exclusively tin scrap, the company is making all preparations also for the use of old cans, provided they are obtainable in sufficient quantity and at sufficiently low cost. Further the process to be used will be entirely different from the one now employed by the company at their Chrome, N. J., plant. Instead of making anhydrous tetrochloride of tin, as is done in Chrome, they will make pig tin in East Chicago.

The buildings for the plant will be completed within a few weeks and it is expected that the plant will be in operation in the Spring of 1912.

Fixation of Atmospheric Nitrogen in Norway.

Arrangements have recently been made, according to *Tidens Tegn*, Kristiania, Norway, between the Badische Anilin and Soda Fabrik and the Norsk Hydroelektrisk Kvaelstofaktieselskab, to the effect that the management of the large airsalpeter developments in Norway will be laid in the hands of the Norwegians, while it has up till now been in charge of the German company. Mr. Sam Eyde, joint inventor with Professor Birkeland of the nitric acid process bearing their name, is to be at the head of the undertaking.

The greater part of the capital interested in the new and old developments is French. The Germans have about one and one-third million dollars, and the Canadians two and two-third million dollars. Swedish as well as Norwegian capital is also quite heavily interested.

The paid-up capital is sufficient to finish the large developments of water power at Rjukan, which is going to be used exclusively in the airsalpeter industry.

NOTES

Unveiling of Bust of Wolcott Gibbs at Chemists' Club.—On the evening of Saturday, Nov. 25, the bust of the late Professor Wolcott Gibbs will be unveiled with appropriate exercises at the Chemists' Club in New York. The bust will be presented on behalf of the Gibbs family by Mr. Geo. Gibbs and will be accepted by Dr. Moore as president of the club. Prof. Charles E. Munroe will deliver the main address. There will be a reception afterwards and a fine musical program has been arranged. The ladies of the club members have been specially invited.

The American Platinum Works of Newark, N. J., have acquired the business of the Croselmire & Ackor Co.

The **Standard Silver Co., Ltd.**, of Toronto, Canada, are building a new five-story factory for the manufacture of silver-plated hollow ware (on Britannia metal, brass, German silver and copper).

The **Geary Water-Tube Boiler** is the subject of a neat and well illustrated little pamphlet of the manufacturers, the Oil City Boiler Works, of Oil City, Pa.

The Care of the Balance.—Messrs. Voland & Sons, makers of precision balances and weights in New Rochelle, N. Y., have just issued a neat little pamphlet on the "use and care of precision balances." It contains elementary but important instructions on the setting up of a balance, when received from the factory, how to place the parts in position, how to obtain the equilibrium, how to clean it, how to release the beam, where to put the balance, how to lubricate, when this should be necessary, etc. There are also some tables on weights and measures and decimal equivalents.

Dredging for Gold is the title of an interesting pamphlet distributed at the recent California meeting of the American Institute of Mining Engineers and containing a well illustrated description of the two newest gold dredges installed by the Natomas Consolidated, of San Francisco and Sacramento. They are of the California endless chain type, having digging buckets of 15 cubic feet capacity. They dig to a depth of 55 feet below the water line of the pond in which they float. They have a maximum theoretical capacity of about 650 cubic yards per hour and have, under favorable conditions, handled 500 cubic yards per hour of running time in continuous operations in formation extremely difficult to handle.

Experiments in slime filtration carried on by George J. Young at the Mackay School of Mines, Reno, Nev., lead him to believe, as stated in a paper presented to the American Institute of Mining Engineers, that: the vacuum filter is best adapted to the treatment of slime containing a large proportion of clayey material; the pressure filter does its best work on slime which contains much coarse and fine sand; while for material containing a small proportion of fine clayey material and much fine sand, either vacuum or pressure filters could be used with perhaps equally good results. Of the vacuum filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Electrolytic Production of Iron.—A recent British patent (28,226, Oct. 19, 1911) of Mr. S. O. Cowper-Coles is mentioned in London *Electrical Engineering*. It relates to an electrolytic process, in which the cathodes consist of thin sheets of electrolytic iron, and the anodes of bars of pig iron of any kind. The electrolyte can be sulphuric or hydrochloric acids, or a mixture of both, or a ferrous ammonium sulphate. Any thickness of iron may be deposited on the cathode, which can be removed from the cell and the iron taken off. To prevent the anodes becoming coated with carbon, graphite or other impurities, brushes are mounted on a spindle, and attached to the cathode. These revolve and keep the anode surface free.

Seger Cones.—In last year's report of the German Reichsanstalt numerous tests of Seger cones in ceramic kilns in practice are mentioned, which were accompanied by thermo-electric measurements. It was found that the Seger cones broke down in the ceramic kilns at considerably lower temperatures than in the electric furnaces of the Reichsanstalt. This may be due to the long time of heating (60 hours), and perhaps also to a chemical effect of the furnace gases. At any rate, this fact must be taken into consideration in using Seger cones for determining temperatures.

Asbestos is most widely used as a fireproof cloth for theater curtains. Other important uses are the making of firemen's clothing, and insulating materials for both steam and electric power plants.

Sealing Wires in Glass Globes.—A method of sealing wires into glass, which, while primarily intended for mercury-vapor lamps, may have wider use, is described in a recent British patent (No. 1880 of Oct. 12) of Mr. G. B. Burnside. As noted in London *Electrical Engineering*, the wire is placed through a small aperture in the glass or other vitreous substance and heated by a blowpipe flame until perfect cohesion between the two takes place. The seal is then removed from the flame, and when at a dull red heat it is cooled by gradually increasing immersions, of about two or three seconds' duration, in some sperm or similar oil or in wax. For currents above 15 amp. round tubular conductors, plugged with a conducting material, are employed. This form is very useful for mercury-vapor lamps or rectifiers, as the tubular shell is made of a substance that does not form an amalgam, and thus protects the copper from the mercury.

Personal

Mr. **Jacques Abady,** a director of the English firm of Alexander Wright & Co., was recently in this country on a two weeks' visit to their American branch, the Precision Instrument Company of Detroit, Mich. In addition to his business activities Mr. Abady is a barrister, the author of Abady's Gas Analyst's Manual, and other works on combustion, and one of the inventors of the Simmance-Abady combustion recorder and the Simmance-Abady vacuum and pressure gauges. His visit had to do with several new inventions which have passed the experimental stage and which will be shortly on the market for general use in steam and gas plants.

Mr. H. Standish Ball, late of McGill University, Montreal, has left for South Africa, where he will accept a position. His address is P. O. Box 2536, Johannesburg, Transvaal, South Africa.

Mr. R. B. Carnahan, Jr., Vice-President of the American Rolling Mill Company of Middletown, Ohio, has returned from a six months' business trip through Europe in the interest of their well-known American ingot iron.

Mr. Arthur H. Carpenter has returned to Denver after an extended stay in Nevada on professional work.

Mr. C. W. Carpenter, of New York, secretary of the Colorado Vanadium Company, recently made a trip of inspection to the company's mines in southwestern Colorado. He was accompanied by Mr. W. F. Bleecker, metallurgist of the company, and Mr. Kirby Thomas, consulting mining engineer.

Mr. J. V. N. Dorr, metallurgist, of Denver, has been appointed general manager of the Mogul Mining Company, Pluma, S. D., for which he has been consulting engineer for some time.

Mr. Arthur Lakes, Jr., of Denver, has gone to Ymir, B. C., to take charge of the mines and mill of the Ymir-Wilcox Development Company.

Mr. James C. Lawrence has resigned his position as chemical engineer in the U. S. Forest Service, to take up the practice of his profession, and has accepted the position as consulting engineer in charge of the operations of the Forest Products Chemical Company of Memphis, Tenn.

Mr. Warren B. Lippincott, who has been identified with the zinc industry for a number of years, has left the Colorado field and gone to Missouri. His partnership with J. M. McClave, of Denver, has been dissolved and his office will be at Joplin, Mo.

Mr. J. B. McIntosh, formerly of Salt Lake City, has removed his office to Los Angeles. He will have charge of the construction of the new copper smelter to be built for the United Verde Copper Company at Jerome, Ariz.

Mr. C. H. Repath, consulting engineer, of Los Angeles, Cal., has charge of the designing of the new copper smelter for the United Verde Copper Company at Jerome, Ariz.

Mr. B. F. Rice, vice-president and general manager of the McKesson Concentrator Company, of Colorado Springs, has opened an office in the Ideal Building, Denver.

Mr. Etienne A. Ritter, of Colorado Springs, has been examining the carnotite deposits in Utah.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

495,637, April 18, 1893, Johannes Pfleger, of Frankfort-on-the-Maine, Germany.

Zinc. Electrodeposits zinc from solutions, or extracts pure zinc from zinc ore, hard zinc, zinc alloys, etc., by use of basic zinc salts obtained by dissolving oxide of zinc in concentrated solutions of sulphates or chlorides of zinc. When used as an electrolyte, all dissolved impurities, such as copper, are precipitated. Like impurities from soluble anodes cannot be dissolved, as basic copper chloride or sulphate is insoluble in the basic zinc solution. The basic zinc salt may be added to neutral solutions; or, with oxide of zinc in suspension, to acid salts. When used as electrolyte, formation of a spongy zinc deposit is prevented. To the solution of basic zinc salt may be added a good conducting neutral salt, such as chloride of sodium or magnesium, of sulphate of potash, and the solution circulated through the electrolytic bath. If necessary zinc oxide may be intermittently or continuously added.

495,715, April 18, 1893, Samuel R. Whitall, of Salt Lake City, Utah.

Gold and silver. Treats chloride, sulphide or arsenide ores containing, besides gold and silver, copper, iron and other impurities, by crushing to pass 16 to 40 mesh screen, then pouring over the dry ore a solution containing ½% to 1½% (of quantity of ore) potassium cyanide, with about 1/3 this quantity of sodium hyposulphite. Water is added in quantity sufficient to cover the ore and allowed to stand until precious metals are dissolved. Gold and silver are then precipitated by zinc or by electrolysis, platinum or carbon anodes being used and a cathode of wires or strips of metal, preferably iron, arranged in parallel lines on copper wires. Gold and silver will be deposited on wires as fine powder, which is collected, dried and melted into bars. Barium, calcium or sodium cyanides may be used instead of potassium cyanide.

If the ore is siliceous, it is first treated with caustic soda and steam under pressure to form a soluble silicate, thereby freeing the precious metals from their combinations with base metals. If the ore is sulphide, it is first given a chlorinating roast with salt and lixiviated with water. Copper or lead contained in the wash water is precipitated by any well-known means.

497,014, May 9, 1893, Frederick W. Cleghorn, of Salt Lake City, Utah, Assignor to Charles G. Brown, of same place.

Gold and silver. The ore is pulverized and screened through a No. 10 screen and saturated in a tank with solution of hyposulphite of soda and sulphuric acid while heated by steam. The sulphuric acid neutralizes impurities of solution. The gold and silver are precipitated as sulphides by calcium or sodium sulphide dried, roasted and cupelled.

The ore remaining in the tank is saturated with sulphuric acid, or muriatic acid, and salt. The gold and silver are precipitated from the solution as chlorides by metallic iron and passing an electric current through the solution. The remaining solutions may be used again.

501,997, July 25, 1893, Stephen H. Emmens, of London, England.

Iron, copper, etc. The apparatus comprises a shallow rectangular vat of paraffined wood. The sides and bottom of the vat are lined with carbon, which forms the anode. An inner rectangular wall of paraffined wood is placed on the carbon floor, leaving a space about four inches wide between it and the carbon side lining on all four sides. The space between the inner wall and the sides of the vat is filled with the ore or other material and a layer of such material about six inches deep is placed on the carbon floor. The electrolyte (such as an aqueous solution of ammonium sulphate or chloride) is poured into the vat until there is a depth of about four inches above the ore, but within the inner rectangular wall. The cathodes consist of a series of rectangular perforated plates suspended parallel with the floor and connected to cross bars extending across the vat, the cathodes being hung about two inches from the layer of ore. The electrolyte can pass underneath the inner wall and saturate the material between the inner wall and the wall of the vat, this arrangement preventing the short-circuiting between the cathode and the ore. The acid constituent of the electrolyte unites with the metal or metallic oxide to form a solution which is in turn decomposed, the metal being deposited and the acid set free to take up a fresh charge of metal from the ore. The perforations in the cathodes facilitate the escape of hydrogen and circulation of the fluid in the upper portion of the vat. When iron ore or scrap iron is being treated for the production of malleable iron, paraffin oil may be poured on the surface of the electrolyte to prevent oxidation. A screen of coarse canvas impregnated with paraffin oil may be supported beneath the cathodes to collect material falling from the cathodes. The apparatus is also applicable to reduction of copper ores, treatment of jewellers' sweepings, sulphides, and other metalliferous substances.

BOOK REVIEWS.

Die Metallurgie des Wolframs.—Praktisches Handbuch von Dr. **Hans Mennicke.** 416 pages, 39 illus. Price 15 Marks in paper cover, 17 Marks bound in cloth. Berlin: M. Krayn.

This book is intended primarily as a reference book. To many it will seem almost incredible that the metallurgy of tungsten requires so bulky a volume, yet the information contained in these pages is for the most part very concise. Although but a few decades ago tungsten was classed as a rare metal, it has in recent years taken a very prominent place among the useful metals. The author has covered every article and patent pertaining to tungsten that have appeared in the last twenty years.

Among the chapter headings are: Ores, Reduction Methods, By-products including scandium, titanium, tantalum and niobium, Manufacture of Sodium Tungstate, Tungstic Acid, Alloys and Compounds, Tungsten Filaments, Fire-proofing Cotton and other Fabrics, and Analytical Methods. Although the chapters on tungsten steel and on tungsten-filament manufacture are comparatively brief and the information based largely on patent literature, the chapters on the production of tungsten powder are particularly well treated, the author having had considerable experience in this line. The book will be welcomed not only by research students but also by steel metallurgists and incandescent-lamp manufacturers.

The Soil Solution. By **Frank K. Cameron.** 125 pages. Price $1.25. Easton, Pa.: The Chemical Publishing Company.

This is an interesting elementary chemical text-book on the nutrient medium for the growth of plants in soil, written from the physico-chemical standpoint. As a rule, books on this subject are either intensely technical and difficult, or they are popular in character so as to aid the practising farmer or horticulturist in raising crops for the market.

This book occupies a middle position. It is neither so technical as to appeal only to specialists, on the one hand, nor so popular and practical as to appeal solely to business men on the other. The book will interest the scientific student of agriculture, the student of applied chemistry and the scientifically minded members of the public at large.

Its thirteen chapters deal with the following topics: The soil, soil management, soil analysis, plant-food theory, dynamic soil phenomena, film water, mineral constituents, absorption by soils, plant growth and concentration, supply and removal of mineral nutrients, organic constituents, fertilizers and alkali.